FALCONBRIDGE

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# IN SITU MONITORING AND COMPUTER MODELING OF A CEMENTED SILL MAT AND CONFINES DURING A TERTIARY STAGE PILLAR RECOVERY

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S. (Stan) Bharti - Superintendent Mines Technical Services and Project Manager
W. E. (Bill) Hedderson - Backfill Engineer (1985, 1986)
P. H. (Phil) Hopkins - Backfill Engineer (1986 to present)
D. M. (Doug) Morrison - Senior Geomechanics Engineer (1985, 1986)
B. M. (Brian) O'Hearn - Geomechanics Engineer
M. (Mike) Beaudry - Engineering Assistant
G. (Gerry) Allan - Blasting Technologist

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# ABSTRACT

This report outlines a diamond drilling technique, developed to extract relatively undisturbed core samples of weakly consolidated hydraulic mill tailings backfill, together with results from in situ and laboratory material quality tests. Work using a numerical model for simulation of an underground backfill structure is also reported.

In the process of this study, backfill coring techniques were developed at Falconbridge's Lockerby Mine. Material quality determinations were then done on the backfill core samples to obtain the value for: Dry bulk density, moisture content, cohesive strength, angle of shearing resistance, uniaxial compressive strength and pressure modulus. The resulting drill holes in the fill were then used to accommodate soil extensometers, geophones and Glotzl cells. These instruments were used to monitor the backfill's behaviour as mining progressed adjacent to the fill. An ultrasonic measuring device was also used to help profile the fill failure that resulted. The data obtained from the material quality determinations and in situ instrumentation was used to numerically model the backfill behaviour when subjected to loading due to adjacent mining.

The results of this work indicate that it is possible to obtain up to 95% undisturbed core samples from even weakly consolidated backfills. It was also shown that, with accurate calibration of presently available numerical models, representative modeling of backfill structures within an active mining environment can be carried out.

# RÉSUMÉ

Ce rapport présente une technique de forage au diamant développée pour extraire des échantillons de carottage relativement intacts d'un remblai hydraulique composé de résidus de concentrateur peu consolidés. On présente aussi les résultats d'essais de qualité des matériaux in-situ et en laboratoire, ainsi qu'une modélisation numérique pour simuler une structure souterraine de remblai.

Au cours de cette étude, les techniques de carottage pour remblai furent mises au point à la mine Lockerby de Falconbridge. Les mesures de qualité des matériaux furent prises sur les échantillons de carottage de remblai, dans le but de déterminer les paramètres suivants: poids volumique sec, taux d'humidité, force de cohésion, angle de résistance au cisaillement, résistance uniaxiale en compression et module de pression. Les trous ainsi forés furent utilisés pour l'installation d'extensomètres, de géophones et de cellules Glotzl. Ces instruments ont alors permis de suivre le comportement du remblai, parallèlement à la progression des travaus miniers adjacents au remblayage. On se servit également d'un appareil de mesure ultrasonique pour aider à l'établissement du profil de rupture qui en résulta. Les données obtenues par les mesures de qualité des matériaux, et par l'appareillage "in-situ", servirent à la modélisation numérique du comportement du remblai soumis au chargement par les activiés adjacentes des travaux miniers.

Les résultats de ces travaux indiquent qu'il est possible d'obtenir des échantillons de carottage jusqu'à 95 pour cent intacts même dans un remblai qui est peu consolidé. On démontre aussi qu'en calibrant avec précision les modèles numériques qui sont présentement disponibles, on peut accomplir une modélisation représentative des structures de remblai dans un environnement minier dynamique.

## SUMMARY

The main purpose of this study was to develop a method for in situ sampling of backfill and use the physical property data and monitoring information obtained to simulate backfill behaviour using computer modelling. The field work was originally scheduled at Falconbridge's Strathcona Mine but was shifted to Lockerby Mine due to ground problems at Strathcona. The project was carried out under a Department of Supply and Services (DSS) Contract Number 0SQ85-00292.

A blasthole pillar at Lockerby Mine (31-176 Stope) was selected for the project as it was being recovered against two backfill walls, approximately 70 meters high, using a vertical crater retreat (VCR) mining method. Diamond drilling, performed by Trow Limited, Sudbury Divison was conducted with a Craelius Diamec 251 drill and a split tube core barrel. Core recoveries with backfill were as high as 95%. The stope was instrumented using extensometers, pressure meters and geophones to study the static and dynamic effects of mining on backfill. The monitoring data, together with the physical property strength information obtained from in situ samples, was used to develop a number of two-dimensional numerical simulations using FLAC (a finite difference computer modelling code) and MUDEC (a distinct element computer modelling code). Both the codes have been developed by the Itasca Consulting Group in Minneapolis, Minnesota, and have been used by Falconbridge since 1986. A large slough of backfill, which unexpectedly occurred during mining of the 31-176 Stope, provided an excellent opportunity to calibrate the computer models with in situ results. Computer model results showed good correlation with in situ fill behaviour.

Finally, a limited number of computer simulations of a sill mat were conducted, using physical property data obtained from Lockerby, to study sill mat behaviour at one of Strathcona's mining blocks. The models indicated that the current sill mats in use at Falconbridge are stable. This aspect requires further investigation.

The results of this study will be of significant benefit to the mining industry in Ontario and the rest of Canada. An alternative technique for undisturbed sampling of backfill has been developed that resulted in higher recoveries. In addition, good calibration was established between the computer models used and in situ fill behaviour. This information will enhance backfill design in Canadian mines and lead to further optimization of placement of backfill.

It is recommended that the Canada Centre for Mineral & Energy Technology (CANMET) consider developing backfill sampling techniques further to make them more cost effective. This will encourage mines to regularly sample backfill and, over time, a large data base on backfill properties, strengths and in situ behaviour can be collected and used in future mine design. Consideration should also be given to broader distribution of the Itasca computer codes (FLAC and MUDEC) because of their capability to accurately simulate backfill behaviour.

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# 1.0 INTRODUCTION

This report presents the results of a study on "In Situ Monitoring and Computer Modelling of a Cemented Sill Mat and Confines During a Tertiary Stage Pillar Recovery." The project was completed for the Department of Supply and Services (DSS) through the Canada Centre for Mineral & Energy Technology (CANMET) under contract serial number 0SQ85-00292. The main objectives of the contract are defined in the statement of work as follows:

1) Review previous investigations of backfill monitoring and backfill failures by literature survey, site investigations and interviews throughout the hardrock mining industry. Analyze previous approaches to the monitoring and computer modelling to guide the proposed research.

2) Establish drilling techniques, probably using a soils engineering sub-contractor to obtain maximum recovery of core while producing a precision hole in the fill for the insertion of instrumentation.

3) Select a test site with adequate access and control.

4) Procure and install suitable instrumentation for the characterization of backfill in situ.

5) Develop and/or adapt computer models to the generally expected range of test values and to the geometry of the test site.

6) Gather data for input to the models. Review data to determine any changes necessary to improve the quality of the input.

7) Validate/calibrate the model and predict the convergences/stresses etc. to be expected during a subsequent planned extraction.

8) Repeat the process of instrumentation and monitoring to demonstrate the capability of the model.

9) Produce a paper for delivery at an industrial seminar appropriate to the topic.

The original field work was scheduled at Falconbridge's Strathcona Mine under a large sill mat. As the project progressed, however, Strathcona experienced serious rockbursting problems and a large proportion of the area scheduled for testing had to be temporarily closed down. In discussions with the CANMET Scientific Authority, Mr. Charles Graham, it was decided to move the initial drilling and testing program to Falconbridge's Lockerby Mine, where pillar recovery against backfill in blasthole stopes was in progress. Pillar mining at Lockerby provided an ideal opportunity to develop appropriate drilling techniques for sampling of backfill and to monitor their static and dynamic behaviour. Once the sampling was completed, the in situ properties and instrumentation data were used to develop and calibrate computer models and eventually simulate in situ sill mat behaviour.

The present state-of-the-art numerical modelling software, the FLAC and MUDEC codes developed by Itasca Consulting Group Incorporated, were used for the numerical modelling. The models were calibrated with instrumentation data and a backfill failure that unexpectedly occurred in the test stope. Finally, a cemented sill mat, of a design recently in use at Falconbridge's Sudbury Operations, was modelled to confirm the mat's structural integrity and demonstrate MUDEC's ability to accurately model cemented sill mats.

## 2.0 CONCLUSIONS

1. A number of drilling techniques were evaluated to assess the most suitable method for obtaining high percentages of undisturbed backfill recovery. The Diamec 251 diamond drill was modified using a low rpm/high torque unit with a split inner tube core barrel to obtain the desired recovery levels. Core recoveries were as high as 95%, even in extremely weak cemented hydraulic backfill. This is a significant improvement over typical core recoveries of approximately 15% using conventional methods.

2. The diamond drilling equipment used is relatively expensive and is unlikely to be used by mining companies on a routine basis. In addition, the operation of the diamond drill requires a highly qualified operator. Further work is, therefore, necessary to develop more cost effective drilling techniques in backfill.

3. The use of a co-polymer flushing agent can significantly enhance overall core recovery when drilling through cemented hydraulic backfill. After testing a number of agents, a granular co-polymer of sodium acrylamides/acrylates, supplied by Westcoast Drilling Services Limited, appeared to be more suitable. In addition, the agent appeared to have minimal effect on the moisture content of the samples.

4. A number of instruments were used to monitor the in situ behaviour of backfill during mining. These included Glötzl cells (flatjacks), soil extensometers, an ultrasonic void measuring device developed by the U.S. Bureau of Mines, a Texam pressure meter used in boreholes, and penetration cones. Most of the instrumentation worked successfully but required careful installation and a qualified technician on the site to monitor the equipment. Future development of instrumentation for cemented backfills should be encouraged and should focus on more rugged equipment at an attractive cost.

5. The physical properties of backfill were evaluated in the laboratory from samples obtained with diamond drilling and through in situ monitoring. The results are summarized below:

| | Strongly Cemented (Tailings to Cement Ratio less than 10) | | Weakly Cemented (Tailings to Cement Ratio over 25) | |
|---|---|---|---|---|
| | In Situ | Laboratory * | In Situ | Laboratory* |
| Pressure Meter Modulus | 400-600 MPa | | 40-200 MPa | |
| Uniaxial Compressive Strength | 2000-4000 kPa | 1050-1600 KPa | 200-1000 kPa | 250-600 KPa |
| Cohesive Strength | 1000-1500 kPa | | 100-400 kPa | |
| Moisture Content | 14.3% | 12.5% | 16.8% | 15.0% |
| Angle of Shearing Resistance | 10-15 degrees | | 35-50 degrees | |
| Dry Bulk Density | 2113 kg/m$^3$ | 1946 kg/m$^3$ | 2113 kg/m$^3$ | 1946 kg/m$^3$ |

* Laboratory results were obtained from Falconbridge lab tests-1983-present

The above results are more representative of in situ backfill than those typically obtained from samples prepared in the laboratory and reflect the variety of cemented backfills that can be encountered underground due to segregation. Fill segregation depends on the placement method, drainage techniques employed, the size of the stopes and the pour density. Further in situ physical property data needs to be obtained at a number of mines in Ontario and Canada to allow adequate assessement of the variability in backfills and to develop a broad data base on in situ fill properties for use by the mining industry.

6. The dynamic effects on backfill were studied by conducting a blast vibration monitoring program as mining was in progress in the 31-176 Stope. Four geophones were installed at approximately 15 meters from the blast and the peak particle velocities (P.P.V.) measured with each blast. Results indicated that P.P.V.'s of up to 53 centimeters per second were encountered at this distance. These velocities are obviously high and the slough of backfill on one side of the pillar may well have been caused, at least partially, by blasting. The impact of blasting

on backfill, and how to minimize it, needs to be studied further in a separate project.

7. Two computer models, FLAC (a finite difference code) and MUDEC (a distinct element code), were selected to simulate static and dynamic fill behaviour during mining. Both these programs have been developed by the Itasca Consulting Group based in Minneapolis, Minnesota and represent the present state-of-the-art in two-dimensional numerical modelling. The programs have been used by Falconbridge over the last three years. The results of the simulation indicated extremely close correlation between in situ fill behaviour and that predicted by the computer model. The MUDEC model was also used to simulate the dynamic effects of blasting on backfill and showed that large peak particle velocities in the backfill may well have triggered the failure that was encountered during mining. It may, therefore, be concluded that current computer modelling techniques are capable of simulating in situ backfill behaviour provided good input data is available.

8. Once confidence was gained in the computer modelling, one of the recent sill mats at Falconbridge's Strathcona Mine was evaluated. Results indicated that the current sill mat design at Falconbridge is acceptable under conditions of static loading. Time constraints did not allow modelling of the effects of dynamic loading on the sill mat, but the ability of MUDEC to simulate such loading was successfully demonstrated by the backfill modelling. Further follow-up work using the results of this study, should be carried out on sill mat design.

## 3.0 RECOMMENDATIONS

1. The cost of equipment and level of operator expertise required for diamond drilling in backfill precludes its use as a routine tool in backfill sampling. Further work, possibly with the manufacturing company, is necessary to develop a reliable and relatively inexpensive method for coring and obtaining fill samples. Since the physical properties of backfill in stopes can vary significantly from those determined in the lab, it is essential that regular in situ sampling of fill be conducted at different mines in Canada. This will allow the establishment of a nation-wide data base for the mining industry for use in future backfill design.

2. Blast vibration damage in stoping appears to be a major cause of dilution from backfill. CANMET should conduct a follow up study on blast design and blasting practices at mines recovering pillars against backfill. The objective of the study should be to determine the optimum explosive charge, drilling patterns and firing sequence so that damage to backfill can be minimized. In addition, consideration should be given to developing methods for reinforcing backfills to better withstand dynamic loading. Since fill dilution is a major problem at most mines, this project should be given a high priority.

3. Further work needs to be carried out with computer models to simulate in situ backfill behaviour and improve calibration with observed fill behaviour. In addition, the present programs used by Falconbridge (FLAC and MUDEC) are proprietary and have to be purchased. CANMET should consider developing and/or obtaining these programs for broader distribution and to enable access for the mining industry at minimum cost.

# 4.0 IN SITU TESTING AND MONITORING OF BACKFILL

This chapter presents the result of all the field work carried out for this study. The in situ testing included experimentation and development of methods for obtaining acceptable core recoveries of undisturbed backfill samples. This drilling and core recovery program was carried out by Trow Limited, Sudbury Division under the supervision of Falconbridge Limited personnel. The test stope was instrumented using extensometers and pressure meters. Geophones were used for blast monitoring in the fill during each production blast. This information formed the basis for follow-up numerical modelling (discussed in section 5.0).

## 4.1 Technical Review

Prior to conducting any field work, a technical review on the current state-of-the-art in drilling and instrumentation in backfill was carried out. This review included a literature search, evaluation of previous work funded by DSS and CANMET on backfill and visits to selected mines in Canada. Since Falconbridge was completing a separate study for DSS under sub-contract to Dome Mines Limited (in situ determination of dewatered tailings fill properties in Ontario mines, serial #06SQ.23440-5-9204) that involved visits to Europe and Australia, key aspects of backfill technology relevant to this study were also examined.

The technical review and site visits confirmed that very little work has been carried out on the in situ behaviour of hydraulic backfills. Most of the studies have either focussed on cemented waste rock fills (such as at Kidd Creek Mines in Timmins, Ontario and Mount Isa in Australia) or have been extrapolated from laboratory tests. A 1980 study by Trow Ontario Limited at Falconbridge's Strathcona Mine (sampling and testing of cemented backfill in underground mines, serial #0SQ79-00113) gave poor core recoveries in backfill and highlighted the need to develop better methods.

The technical review confirmed CANMET's assessment on the need to conduct more research in this area of backfill.

## 4.2 Test Site Selection

Careful selection of a suitable drill site within the five operating underground mines of Falconbridge's Sudbury Operations was essential in order to reach drilling objectives.
The following parameters were considered in site selection:

- Accessibility with drilling equipment.
- Direct access to exposed cemented backfill.
- Opportunity to drill in varying cement: tailings ratios.
- Access to sequential, rather than single bulk, backfill pours.

With these criteria in mind, the 31-176 stope overcut (O/C) on Lockerby Mines's 2950 level, was chosen as the drill site (Figure 1). Falconbridge's Lockerby Mine employs the blasthole mining method and produced about 612,000 tons of nickel-copper ore in 1987. The mine has been in production since 1978 and currently obtains a large proportion of its tonnage through pillar recovery against cemented backfill (cement to tailings ratios vary from 1:12 to 1:32). Further details of Lockerby Mine are found in Appendix A.

This specific drill site is situated between two previously backfilled stopes (Figure 2), 31-172 and 31-179 with the following relevant parameters:

| | 31-172 | 31-179 |
|---|---|---|
| Backfill: cement | 12:1 | 16:1 |
| Designed strength (MPa) | 0.69 | 0.46 |
| Date of Pour | 1982 | 1985 |

Lockerby's 31-176 stope was ideally suited to conduct the drilling and monitoring because it provided backfill exposures (Figure 3) and drilling sites within the project time frame. The data could then be used to conduct detailed computer modelling for sill mat design.

## 4.3 Drilling and Core Recovery in Backfill

### 4.3.1 Drilling Objectives

Present technology for drilling in cemented backfills to obtain undisturbed backfill samples and produce an accurate drill hole for instrumentation and monitoring is unacceptable. Typical core recoveries, using conventional diamond drilling methods, average 15% or less. Since in situ hydraulic backfill in stopes can vary greatly from that in the lab, in situ sampling of this material is essential to accurately quantify backfill behaviour in underground mine design. A reliable estimate of in situ material properties such as cohesion, density, stiffness, etc. could lead to further optimization of backfill design and a possible reduction in cement usage and overall backfilling costs.

One of the main drilling objectives is undisturbed sample recovery to enable the determination of key in situ material parameters of the backfill; these parameters are required to accurately incorporate backfill in numerical models. Although some of these values are available for various cemented fills from previous lab work or literature, they can fluctuate greatly from mine to mine. In addition, the placement and dewatering method used for cemented hydraulic backfill, leads to inhomogeneity and different settling rates, coarse and fine particle segregation, and varying cement content ratios. In situ undisturbed sampling and monitoring of fill behaviour are therefore essential.

Undisturbed samples provide a very valuable visual picture of in situ backfill. Although it is accepted that segregation occurs in most backfill pours (due to the method of pouring, draining, etc.), the actual pattern of this segregation is not well understood.

Segregation and cement content affect the economics and safety of fill and a better understanding of how this occurs in cemented backfill can be of value in future mine design.

The extent to which in situ backfill in mines can be monitored as mining progresses is somewhat limited because it is difficult to drill accurate holes for instrumentation. A key objective to the drilling program was also, therefore, to drill accurately in backfill.

### 4.3.2 Drill Selection

Drill selection for this project was based on field experience, both external and at the Falconbridge mines, and recommendations from diamond drill contracting firms familiar with this type of drilling.

In the past, Falconbridge has attempted to recover undisturbed fill samples using conventional diamond drills with single, double and split tube core barrels. None of these methods provided the drilling control required for suitable sample recovery. In many cases, recovery was less than 15% and, in other cases where recovery was higher, the samples were badly broken and/or often saturated with drilling fluid. With this experience, it was determined that the following drill performance criteria were required:

- Accurate control of drilling rotation in the 50 to 200 RPM range.
- Minimal sample contact with drilling fluids.
- Drill barrels that allow for easy core removal.

Discussions were held with various contract drilling firms. Talks with Craelius Division of Unicorn Abrasives of Canada Limited resulted in the final selection of the drill to be used. The following are the basic details of this unit. Specific drill data is contained in Appendix B.

Selected Drill - Diamec 251 (modified)
- Diamec 260 low RPM/high torque rotation unit.
- Diamec 260 rod holder
- Skid type frame
- 30 KW 550 V electric motor
- Craelius T6S 86 x 1500 mm core barrel (double tube, swivel head type)
- Split inner tube core barrel
- Surface set diamond drill bit - 20 SPC

This unit is also portable and easily assembled (Figures 4-6). The power unit and drill frame are quickly detached and easily transported underground in either a standard size cage or scooptram.

### 4.3.3 Drilling and Drill Core Recovery

#### 4.3.3.1 Drill Hole Locations

Six boreholes in total were completed and varied in length from 5.9 m to 8.5 m (Appendix C). The holes were orientated as shown in Figures 7 and 8. The first of these boreholes was drilled solely to optimise drilling rotation, feed pressure, and feed rates, and was not used for sampling or instrumentation purposes. All boreholes were drilled horizontal or upgrade to facilitate drainage of the drilling fluid, and in doing so, minimize the effect of drilling fluids on recovered core samples.

### 4.3.3.2 Drilling Parameters

In addition to the general drill data for the Diamec 251 Drill mentioned in Section 4.3.2, the following are the specific drill settings used:

Feed Load - 0.5 - 0.75 tonne
Water Pressure - 200 - 400 kPa
Feed Rates - 15 - 25 mm/min.
Rotation - 50 - 150 RPM

These settings were varied within the ranges listed above in order to provide optimum penetration with minimum "flushing" requirements for acquiring undisturbed core samples in both weak and strong cemented fills. The use of experienced drillers allowed for adjustments to these drilling parameters while drilling was in progress, and as different strength fills were encountered. These parameters were also varied for deeper boreholes.

### 4.3.3.3 Flushing Agents

Flushing agents used in drilling are important as they can affect the moisture content and physical integrity of core samples. These factors will, in turn, affect the physical properties of the cemented fill samples under examination. In order to minimize this sample deterioration, the following flushing methods were evaluated:

- Pressurized air
- Pressurized water
- Co-polymer mixed with pressurized water

The final flushing agent used during drilling operations was a concentrated, granular co-polymer of sodium acrylamides/acrylates as supplied by Westcoast Drilling Supplies Limited. This solution appeared to work well and core samples were extracted fully intact in even the weaker cemented fill. Although it is difficult to determine the effect of the use of this co-polymer on a sample's moisture content, it is felt that the increase in moisture level was minimal.

### 4.3.3.4 Sample Recovery and Handling

In order to obtain maximum core recovery of undisturbed cemented fill, inner split tube and standard NW solid tube (both using double tube core barrels) proved effective. Both core barrels worked equally efficiently in drilling and sample retrieval, although the split barrel design of the Craelius core barrel provided easier logging and handling of recovered samples.

Samples were placed on sample trays, sealed (to maintain their moisture content) and carefully transported to surface for analytical analysis (Figures 9 and 10). In the weakly cemented samples the core had to be handled carefully in order to maintain the samples in an undisturbed state.

### 4.3.3.5 Continuous Flight Augers

Backfill sampling with flight augers was also tried. This backfill sampling technique has the following advantages and disadvantages:

Advantages - No flushing agents required and thus moisture contents of samples remain unaltered.
- Faster drilling.
- Less expensive.
- Good hole tolerances for instrumentation.
- 100% sample recovery.

Disadvantages- Samples are fully disturbed
- Will not drill through rock, either in fill, or to access fill.

This method for cemented fill sampling was done using the same basic equipment as outlined in Section 4.3.2 and should be considered where sample moisture content, density, grain size distribution, and economical hole sizing for instrumentation are of primary importance (Figure 11).

### 4.3.4 Instrumentation

#### 4.3.4.1 Selection

The following is a summary of the instrumentation that was selected for this project and the purpose for which it was selected.

## SUMMARY OF INSTRUMENTATION

| INSTRUMENTATION | NO. OF UNITS | INSTALLED BY | PURPOSE |
|---|---|---|---|
| Geophones * | 4 | Falconbridge | PPV<br>Blast Wave Frequency |
| Glötzl Cells | 4 | Falconbridge | Fill Loading |
| Soil Extensometers | 3 | Falconbridge | Fill Failure Profile |
| USBM Ultra Sonic Measuring Device | 1 | Falconbridge | Fill Failure Profile |
| Texam Pressuremeter | 5 | Trow Limited | Modulus of Elasticity |
| Penetration Cone | N/A | Trow Limited | Fill Strength |

* Discussed in Section 4.4 - Blast Vibration Monitoring Summary.

### 4.3.4.2 Installation

Glötzl Cells: Glötzl cells (flat jacks) were installed in the 31-179 (backfilled) stope on both the 29-1 sub-level and 3125 level horizons (Figures 12-13). One hole was drilled into the center of the fill at each location with two cells being installed in each hole, one vertically and one horizontally.

Soil Extensometers: Three soil extensometers were installed from the 29-183 crosscut and varied in length from 30 m to 44 m (Figures 14-15). Three to four anchors were installed (as shown) in each hole.

USBM Ultrasonic Measuring Device: This unit (Figure 16), on loan from the USBM (United States Bureau of Mines), was lowered, in increments of 3 meters, into the void created by mining 31-176 stope. This operation was performed for the top 25 m of the stope and a profile of the void taken at each horizon.

**Texam Pressuremeter:** This pressuremeter was used along the entire length of the five major cored holes outlined in Section 4.3.3.1. Unlike the Glötzl pressure tests, the readings obtained at these locations were sets of single readings. Once the individual hole was tested, the instrumentation was removed.

**Penetration Cone:** These tests were attempted on the exposed fill face at several locations in the 37-176 stope overcut (Section 4.2). Due to the stiffness of the fill and the light-weight nature of the Diamec 251 Drill, initial trials resulted in damaged and bent rods. These tests were discontinued at this point.

### 4.3.5 Physical Property Evaluation

#### 4.3.5.1 Pressure Meter Results

Glötzl pressure cell results (Figures 17-18) indicate that a negligible amount of fill loading (in both vertical and horizontal directions) occurred after the adjacent 31-176 stope was mined out. This appears to support the theory that, because rock is extremely stiff compared with backfill, significant load transfer does not occur until the fill has undergone considerable compaction, which would perhaps require something of the order of 15% wall convergence. Detailed Texam pressure meter results are listed in Appendix C. A summary of the results, used in numerical modelling are listed below:

| | Strongly Cemented (T:C < 10) | Weakly Cemented (T:C > 25) |
|---|---|---|
| - Pressuremeter Modulus (EM) | = 400 - 600 MPa | 40 - 200 MPa |
| - Pressuremeter Limit Pres. (P) | = 10 - 20 MPa | 5 - 15 MPa |
| - Em/P Ratio | = 30 - 40 | 8 - 15 |

### 4.3.5.2 Soil Extensometer and USBM Ultrasonic Measuring Device

Results from the extensometers (Figures 19-21) and the USBM Ultrasonic Measuring Device provided the necessary details to map the outline of a fill wall failure which occurred during mining. This profile was compared to a numerically generated failure surface to calibrate the computer model (see Section 5). The void profiles are shown in Appendix D.

### 4.3.5.3 Drill Core Mapping

An example of the core log is shown in Figure 22. This mapping provides a visual record of the fill segregation and quality, and the core itself provides undisturbed fill samples for laboratory determination of physical properties. The complete core mapping is in Appendix E and indicates the widely varying size distributions and cement contents.

### 4.3.5.4 Laboratory Evaluation and Test Work Summary

Cemented fill core samples were subjected to various laboratory tests to obtain the in situ physical properties of the material. This work was carried out by Trow Limited on sub-contract. The following is a list of the laboratory tests and a summary of all test work results.

Laboratory Tests - Uniaxial compressive tests
- Triaxial compressive tests
- Moisture contents
- Cement determination tests
- Dry bulk density

| Test Work Result Summary | Strongly Cemented (T:C < 10)* | Weakly Cemented (T:C > 25)* |
|---|---|---|
| - Uniaxial compressive strength (Q) | 2000 - 4000 kPa | 200 - 1000 kPa |
| - Cohesive Strength (c) | 1000 - 1500 kPa | 100 - 400 kPa |
| - Moisture Content (%) | 14.3 | 16.8 |
| - Angle of shearing resistance (φ) | 10 - 15° | 35 - 50° |
| - Pressuremeter modulus (EM) | 400 - 600 MPa | 40 - 200 MPa |
| - Pressuremeter Limit pressure (P) | 10 - 20 MPa | 5 - 15 MPa |
| - Em/P | 30 - 40 | 8 - 15 |
| Dry bulk density | 2113 Kg/M$^3$ | |

* T:C is the tailings to cement ratio

## 4.4 Blast Vibration Monitoring

### 4.4.1 Monitoring Objectives

Blast vibration monitoring was carried out to determine the following:

- Peak particle velocity (PPV)
- Wave frequency
- Blast duration time

The blast vibration monitoring parameters were used in subsequent numerical modelling to assess the impact of dynamic loading on backfill.

### 4.4.2 Equipment and Installation

Four geophones were installed in total, two on 29-1 sub-level and two on 3125 level (see Figures 23-24). On each of these horizons, one geophone was installed in each of the two backfilled stopes, 31-172 and 31-179, adjacent to the pillar to be mined. The installation locations were chosen to allow comprehensive blast vibration monitoring around the 31-176 pillar.

A Racal 7-channel FM tape recorder and/or two Instantel DS-200 collector units were used to record blast data.

### 4.4.3 Monitoring Results

From August 22 to October 10, 1986, the first ten blasts of 31-176 VCR stope were monitored. The following values, used in numerical modelling, were obtained.

| | |
|---|---|
| - PPV * | 53 cm/second |
| - Dominant Frequency | 30 Hz |
| - Blast Duration * * | 15 MS |

* Measured 15 m from the blast

* * Blast duration for the hole adjacent to the fill wall

Details of this work are given in Appendix F.

# 5.0 NUMERICAL MODELLING

## 5.1 Introduction

Data collected from field instrumentation, Falconbridge files, literature, and work by Trow Limited have been incorporated into input data used to simulate the response of fill to adjacent mining. Numerical models were utilized in the examination of the effects on a cement sill mat and confines during tertiary stage pillar recovery.

An investigation was carried out using FLAC, a finite difference code, and MUDEC, a distinct element code. FLAC was used in the initial stages of modelling to simulate the backfill as a continuum material. This preliminary work gave direction in scoping some input parameters used in MUDEC, which was extensively used throughout this analysis. The MUDEC code was used to simulate the backfill as a discontinuum material.

It was originally planned to monitor the ground movement around a sill mat during the mining of an adjacent stope. However, Falconbridge's mining schedule precluded the availability of a suitable mat which would fall into the project schedule.

A test site was chosen at Lockerby Mine as described in Section 4.2. The recovery of a tertiary pillar mined by the vertical crater retreat method (V.C.R.) was monitored. The philosophy in choosing such a test site was based on the premise that if a tertiary pillar recovery could be modelled successfully, then a sill mat could equally be simulated.

The instrumentation around 31-176 stope would verify the response of the MUDEC model. The successful response of the model to duplicate observations made underground would give us the confidence to try to simulate the reaction of a sill mat to the mining being carried on around it.

## 5.2 The Numerical Models

Flac is an explicit finite difference code which solves the equations of motion in difference form. "Explicit" refers to the nature of algebraic equations used in the numerical solutions. In the explicit method, all quantities on one side of all equations are known, and each equation is simply evaluated to produce the results on the other side. Explicit formulations differ from implicit formulations, ie. boundary elements, where known quantities exist on both sides of the equation. Implicit problems require the solution of simultaneous equations by some technique such as Gauss elimination.

The stope and confines to be simulated are divided into a number of elements with intervening grid points as in any continuum differential code, such as the finite element code. The equations of motion are then solved in a time stepping algorithm at each grid point. For each step in time, the out-of-balance forces at each grid point are determined. These, in turn, are used to determine the velocities at each grid point and the associated strain components. These strains are used in the stress-strain law chosen for the particular zone to determine the corresponding stress components. This cycle is repeated until the user is satisfied that an equilibrium state has been reached.

MUDEC is an explicit, time marching distinct element code which also solves the equations of motion in difference form. Unlike finite elements, distinct elements may interact with any of the other elements and may experience large-scale rigid body translations and rotations. The geometry of distinct elements is defined by the spacing and orientation of joints in the material mass being modelled, with each element corresponding to an individual block of material.

The code is based on force displacement relations which specify the forces between blocks, and a motion law which specifies the motion of each block due to unbalanced forces acting on the block.

Numerical integration of Newton's Second Law is used to determine translation of each block and rotation of each block about its centroid.

The distinct element method has three distinguishing features:

1) The rock mass is simulated as an assemblage of blocks which interact through corner and edge contacts.

2) Discontinuities are regarded as boundary interactions between blocks; joint behaviour is prescribed for these interactions.

3) The method utilizes an explicit timestepping (dynamic) algorithm which allows large displacements and rotations and general non-linear constitutive behaviour for both the material mass and the joints.

In summary, the FLAC code was used in the preliminary analysis to investigate the importance of induced stresses and gravity on the cemented hydraulic fill, as modelled as a continuum. The MUDEC code enabled the effects of blasting to be simulated enhancing its significance on the timing and mechanism of failure on fill.

### 5.3 Summary of Test Site Data and Observations

Figure 28 shows the location of the test stope 31-176. The dimensions of the stope are approximately 11 meters wide by 14 meters on transverse longditudinal with a height of 58 meters. The stope was mined from late August until late October 1986 by the V.C.R. method. A total of 14 cuts were taken including the blasting of the sill. The length of time between lifts varied from two to seven days.

The mining of stope 31-176 left the adjacent backfilled stopes unsupported to a height of about 70 meters. The west stope, 31-172, was filled in 1981 with 12:1 tailings to cement, and the east stope, 31-179, was filled in 1984 with 22:1 fill. During mucking of the stope, a significant amount of fill was noted by scooptram operators in their buckets. Based on these accounts of loose fill in the muck pile, it became evident that fill was peeling off the sidewalls.

The extensometers provided a first estimate of the spacial extent of the fill failure of the east sidewall. Figures 29 and 30 illustrate the magnitude and locations of movement in the x and y directions. It was estimated that the x = 0.025 meter contour indicated the failure surface. As a means of checking the accuracy of this estimate, a sonic void measuring device was lowered into the stope from the overcut (Appendix D). Only the top one third of the stope was surveyed; however, results of the survey showed that the x = 0.025 meter contour aligned reasonably well with the limited survey results. It also showed that the extent of fill failure from the east sidewall was more than double that of the west. On average, the surveyed location of the blastholes showed that the nearest row of holes to the east sidewall was 1.5 meters while the nearest holes to the west sidewall was 2.0 meters. Also, the east stope had ± 3.0% less cement in its fill than the west stope. Based on the blasting pattern and the cement contents of the fill, it seems not surprising that 31-179 failed more extensively than 31-172.

Of note in Figure 29 is the shape of the failure (0.025 m) surface. It is largely vertical near the top of the stope while at the bottom it curves towards the undercut. Such a surface suggests an initial slump at the bottom which progressed horizontally to a vertically orientated weakness plane extending to the surface. The large displacement of the initial slumped material removed the footing that gave direct support to overlying fill.

## 5.4 Preliminary Analysis

As outlined in Section 4.3.5.1 pressure cell results were obtained (Figures 17-18) and showed that little change in load occurred during and after the mining of 31-176. No change of load indicates that the fill was not transferring any load from the adjacent pillars. This phenomenon was also noted by Sinclair et al (1981) who noted that because the rock is very stiff compared with the backfill, transfer of stress is negligible. This is valid until the fill experiences significant compaction, requiring perhaps 15% wall convergence, which never occurred in this instance.

A FLAC model simulating stope 31-179 was examined under static loading conditions. A Mohr-Coulomb constitutive relation with and without tension cut-off was chosen. FLAC also accommodated an "interface". The interface allows grids on either side of it to slide past one another. Thus, for example, a slip plane could be simulated. The interface was situated at the rock fill interface furthest from mining.

Results of the analysis showed that without the tension cut-off a massive slump type failure progressed well into the fill stope. This failure surface was very different from the skin of fill that peeled off in 31-179 stope. However, the tension cut-off model produced a failure that extended ± 2 m into the fill stope and extended up much of the height of the stope. The surface observed in this failure was very similar in shape to that observed underground. In both cases examined, the entire collapse occurred after several cuts had been made. Observations underground showed that fill collapse occurred on several different occasions following production blasting. Thus the model failed to get the timing of the collapse right. However, this timing is strongly related to the blasting schedule and this analysis was strictly a static one.

The preliminary analysis indicates that boundary effects caused by stress transfer through the fill pillar are of little significance thus making gravity important. A Mohr-Coulomb constitutive model with a tension cut-off successfully reproduced the failure surface and further, indicated the peeling nature of the exposed fill wall.

### 5.5 Mudec Simulation

Based on the shape of the failed surface and, because backfill physically peeled away or separated from the fill mass, the distinct element code MUDEC was chosen to simulate the fill failure. The blocks indicated in Figure 31 represent 39-179 stope. The model has dimensions 76 meters x 14 meters. The larger blocks represent an area 4 meters x 2 meters. Two columns of blocks have diagonals through the corners and have been further discritized by splitting the blocks in half. The leftmost column of blocks have been assigned material properties equivalent to Falconbridge ore. The remaining eight columns of the model have been assigned material properties equivalent to the 22:1 backfill which is in place. Backfill in contact with, and adjacent to, the ore interface has been finely discritized in order to allow the failure surface to form its own course with as little dependence on block size as possible (failure can only occur along block interfaces).

The material properties used within the model were established by corroborating measured in situ values with data published in literature. Figure 32 lists all the input parameters pertaining to material properties. If a shear plane along a joint causes two blocks to separate from one another, the joint cohesion, joint friction, and joint tensile strength change. These changes can be found in Property Two of Figure 32.

Boundary conditions consisted of fixing the velocities in the x direction equal to zero on the right vertical boundary. The y velocity was fixed at zero at the bottom horizontal boundary. By fixing these two boundaries at zero, the model simulates the effect of fill resting against a solid rock interface. The other two boundaries were made viscous, allowing shock waves to pass through.

MUDEC simulates a dynamic event like a production blast by applying a force or a peak particle velocity to a given point or area in a sinusoidal manner. The duration of the blast and the frequency of the wave are additional data required to describe the event. By recording the actual seismic pulses resulting from the blasts, with the aid of geophones and vibration analysis equipment, reliable data pertaining to this site was extracted (see Section 4.4 and Appendix F).

The simulation of the mining consisted of the following steps. A 4 metre x 2 metre block was removed to simulate the undercut, Figure 33. A blast was simulated in the two blocks immediately overlying the undercut, Figure 34. The model was then cycled for a period of time such that the effects, ie. ground movement, created by the blast, were fully realized. These two blocks were then deleted from the model (Figure 35). This deletion simulates the extraction of the blasted ore. Again, the model was allowed to adjust to the new conditions. Another blast was repeated in the next two blocks overlying the previous two. This simulation was continued until the top of the model was reached. At this stage, the simulation was stopped. The resulting effects are shown in Figure 36.

## 5.6 Discussion of Modelling Results

Figure 36 illustrates the model after backfill blocks have peeled away from the exposed fill sidewall. The blocks left standing are those in stable equilibrium.

Figure 37 compares the measured failure surface to the predicted surface computed by the numerical model (Figure 36). The two failure surfaces are similar in shape and in dimensions. The MUDEC model was able to reproduce the failure that was known to occur in a well monitored backfilled stope adjacent to a full production stope.

The blasting simulation demonstrated a shock wave propagating outwards from the source location in the form of velocity vectors. After only a couple of meters of propagating outwards some of the wave is reflected back towards the open stope. The reflected wave creates planes of tension between adjacent blocks causing them to separate and peel off into the open stope. The timing of such a collapse occurred immediately after each blast. Consequently, not only the shape of the failure surface was reproduced, but the timing of its occurrence was also duplicated.

## 5.7 Sill Mat Model

A section of Falconbridge's Deep Copper Zone at Strathcona Mine has been used to illustrate the following example. The Deep Copper Zone is located below the Strathcona Mine main ore body. The ore zone has been folded and dips at 60° near the top and 35° near the bottom. It averages 400 meters in strike length. Mining widths vary from 5 to 15 meters.

Figure 38 illustrates, in a transverse section, the geometry of a MUDEC model simulating cut and fill mining in a 5 meters wide stope at a depth of 1200 meters. The timberless sill mat in the model consists of a 2 meter bed of 8:1 fill followed by a 2 meter bed of 16:1 fill which overlie the final two cuts of ore from the sill. The remainder of the fill is 30:1.

The two remaining cuts were extracted one at a time, Figures 39 and 40. Figure 40 shows the principal stresses at equilibrium. The stress vectors have been scaled in length proportional to the longest one, located 5 meters into the hanging wall and measuring 97 MPa. Stresses in the fill measure 0.2 MPa (in the 30:1 fill) and the maximum wall convergence is about two percent. The 8:1 fill beam only deflects several centimeters and, more importantly, remains stable with regard to gravity loading.

In the model, the 8:1 fill was given a friction angle of 36° and cohesion of 0.5 MPa. Time has not permitted a thorough investigation of the sill mat stability for varying friction angles and cohesion. However, critical friction angles and cohesions could be determined and related to cement:tailings ratios. The impact of seismic loading could additionally be examined with such a MUDEC model.

With mining reaching to greater and greater depths, the need to aid our design with predictive models is becoming more evident. We are no longer able to draw on past experience. Stresses have increased and, in some cases, mining induced seismicity must be accounted for in a thorough design. This illustrative example, along with the previous back analysis, demonstrates the capabilities of current software.

## 6.0 IMPLICATIONS FOR THE MINING INDUSTRY.

The results of this study are likely to be of significant benefit to the mining industry in a number of key areas:

1. Core recoveries in weakly cemented hydraulic backfills have been improved significantly. With the techniques developed and established as a result of this project, drilling programs can be performed with a high degree of confidence and good undisturbed samples of in situ backfill can be obtained for laboratory testing.

2. The diamond drilling equipment is also capable of drilling accurate long holes in backfill. Instrumentation can, therefore, be placed in these boreholes and fill behaviour monitored as mining progresses. Although instrumentation in backfill has been used in the past, this is one of the few projects where a detailed instrumentation program for backfill monitoring has been conducted.

3. Blast vibration monitoring results demonstrated that fill behaviour, under dynamic loading conditions, can lead to sloughing and in stability of the fill walls. Blasting, obviously plays a key role in fill stability and this entire aspect needs to be examined. The mining industry should reevaluate blasting methods during pillar recovery against backfill.

4. Numerical modelling techniques historically have not been able to reliably simulate backfill behaviour in situ. With the development of new distinct element codes, such as MUDEC, both the static and dynamic behaviour of fill can be reliably simulated. The two-dimensional MUDEC results are in good correlation with in situ fill behaviour. These, and similar programs, can now be acquired by the mining industry and, provided reliable input data is available, used for evaluating fill stability and behaviour in future mine design.

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M. Scoble, L. Piciacchia, J.M. Robert, "In Situ Testing in Underground Backfilled Stopes", in C.I.M. Bulletin (Canada, 1987), p.33.

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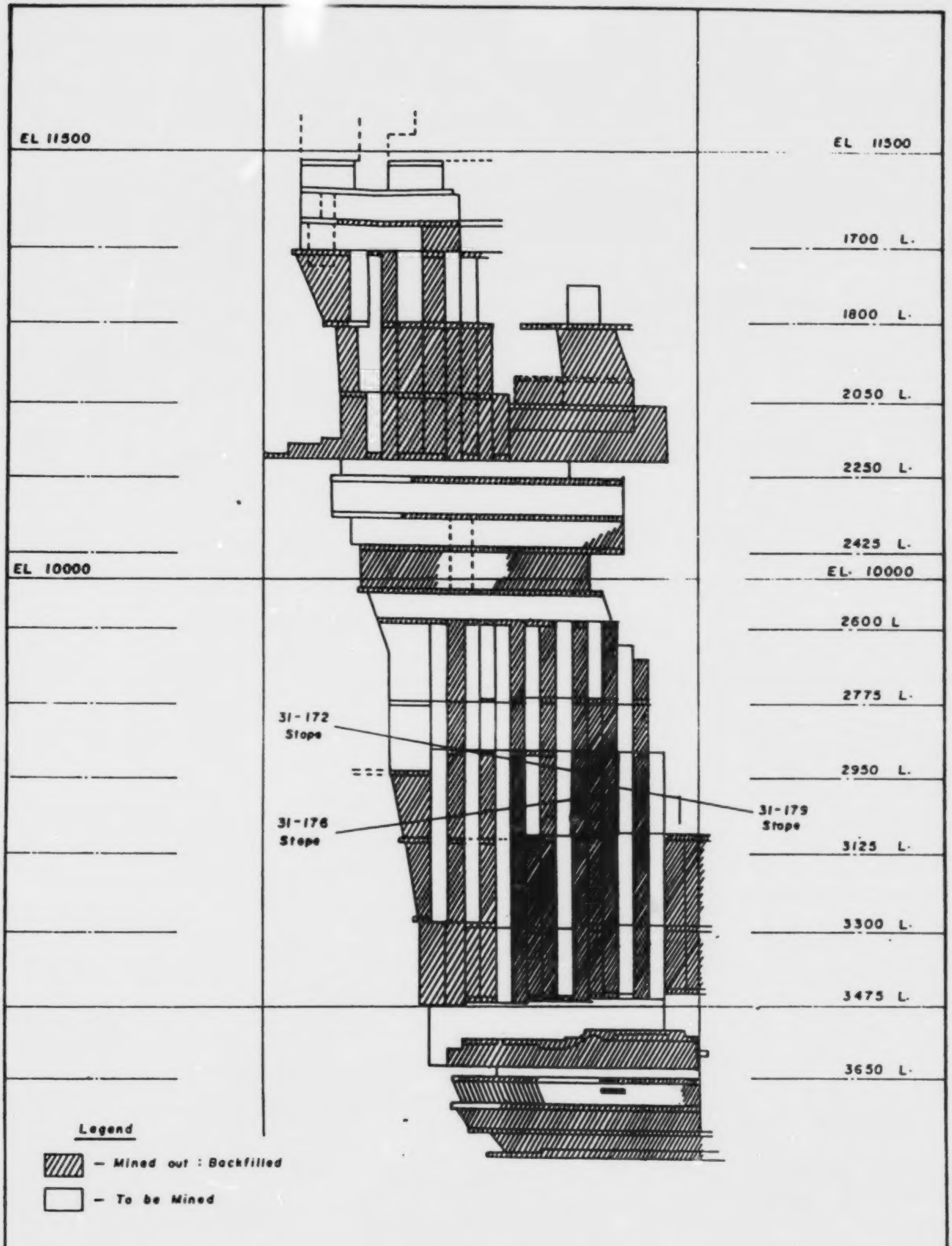
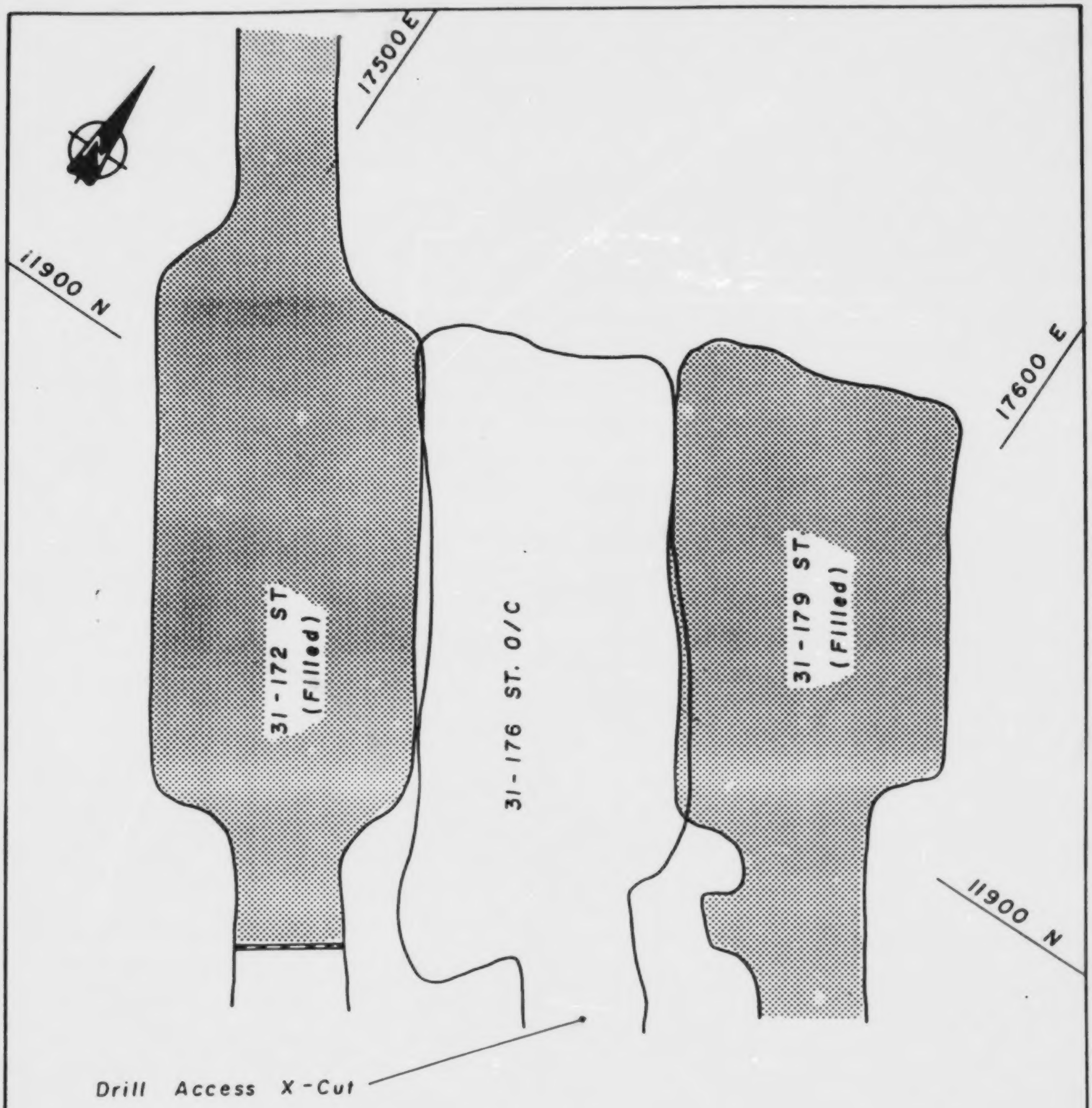

EL 11500
EL 11500
1700 L.
1800 L.
2050 L.
2250 L.
2425 L.
EL 10000
EL. 10000
2600 L
2775 L.
31-172 Stope
2950 L.
31-176 Stope
31-179 Stope
3125 L.
3300 L.
3475 L.
3650 L.
Legend
▨ — Mined out : Backfilled
☐ — To be Mined

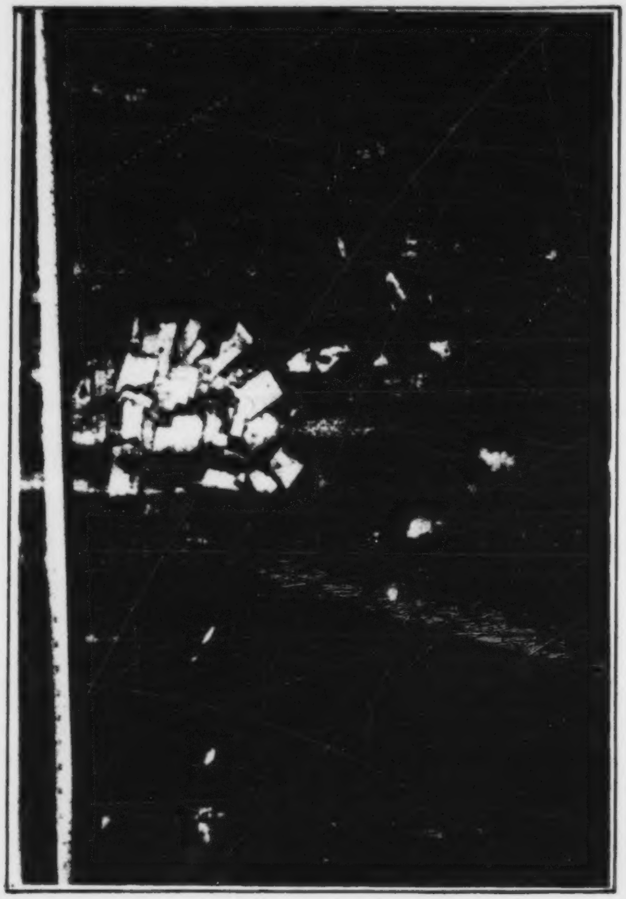
Longitudinal Section of Lockerby Mine

FIG : 1


17500 E
11900 N
17600 E
11900 N
31 - 172 ST (Filled)
31 - 176 ST. O/C
31 - 179 ST (Filled)
Drill Access X-Cut

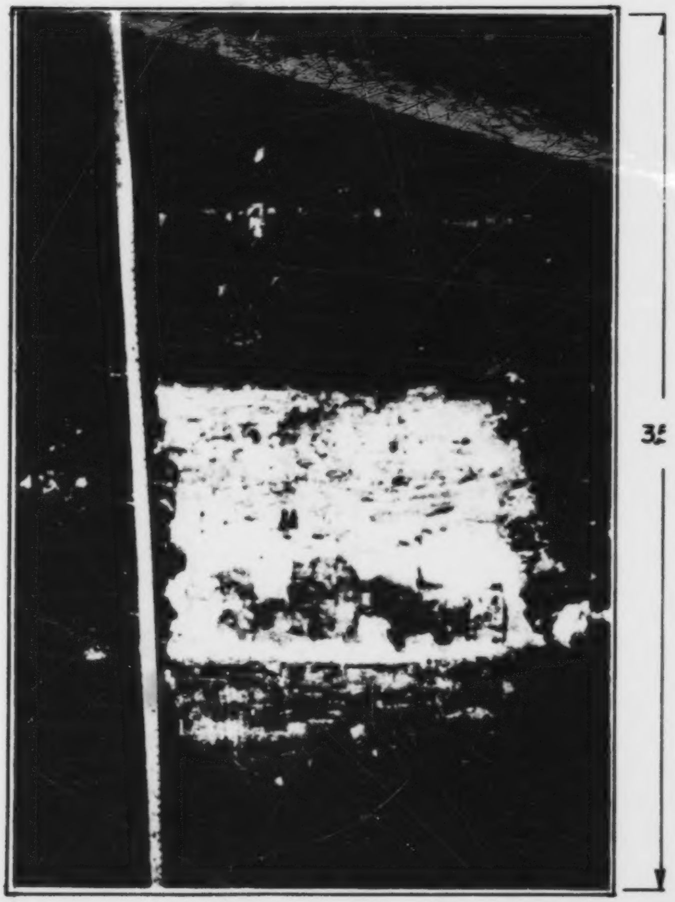
31- 176 Stope — Plan View

FIG : 2



Center of Stope

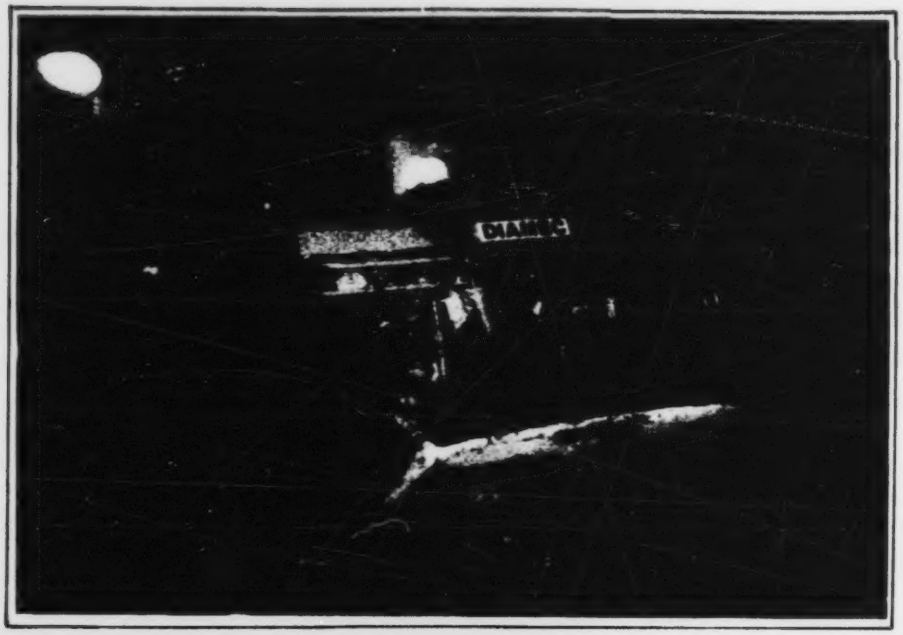
North End of Stope



In-Situ Mine Fill Stratification

31-172 Stope

Figure 3

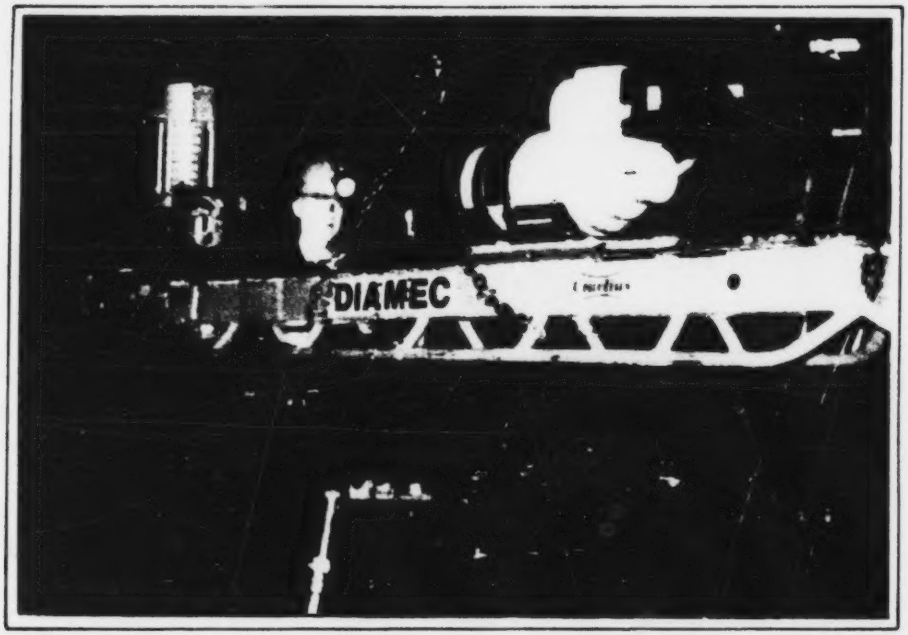


Craelius 251 Drill Unit

Model 30 E Power Unit

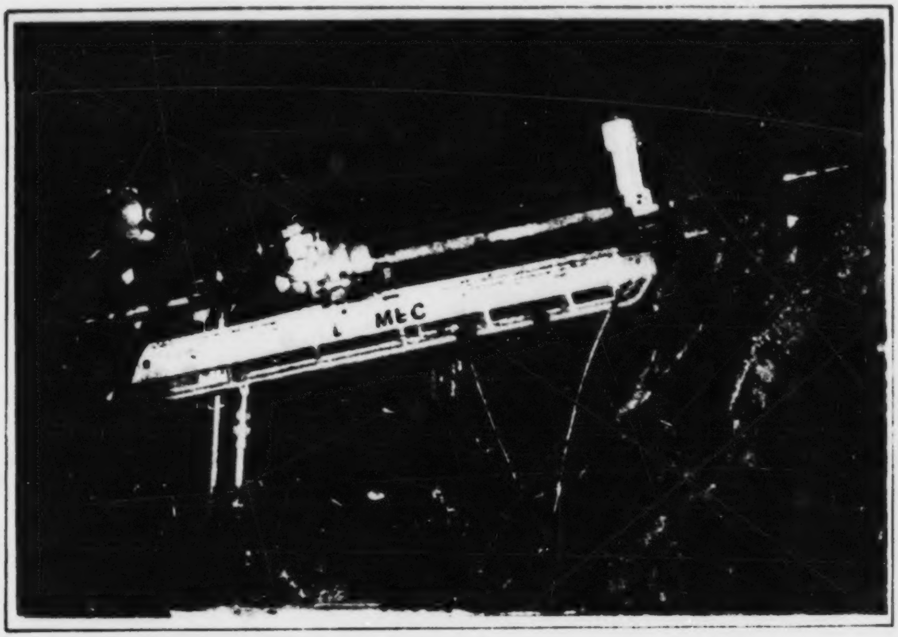
Underground Transportation of The Craelius Drill

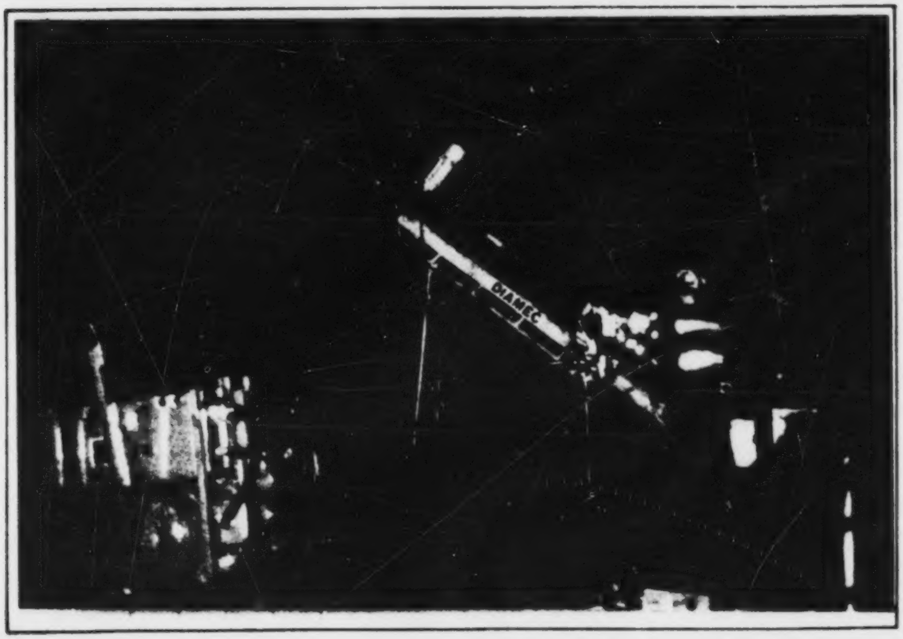
Figure 4



Assembly of Drill Feed Frame

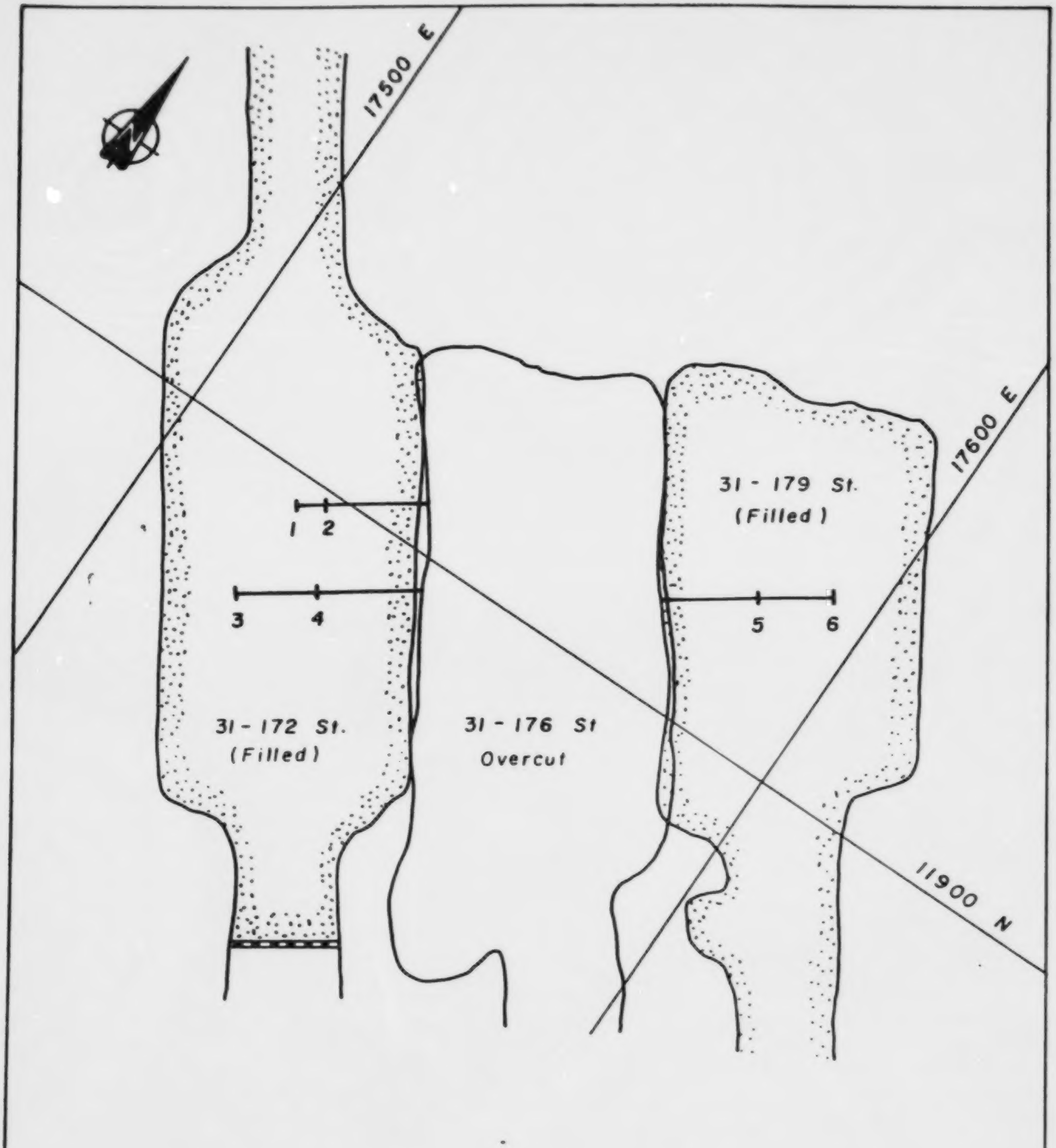
Figure 5





Typical Drill Set Up and Operation

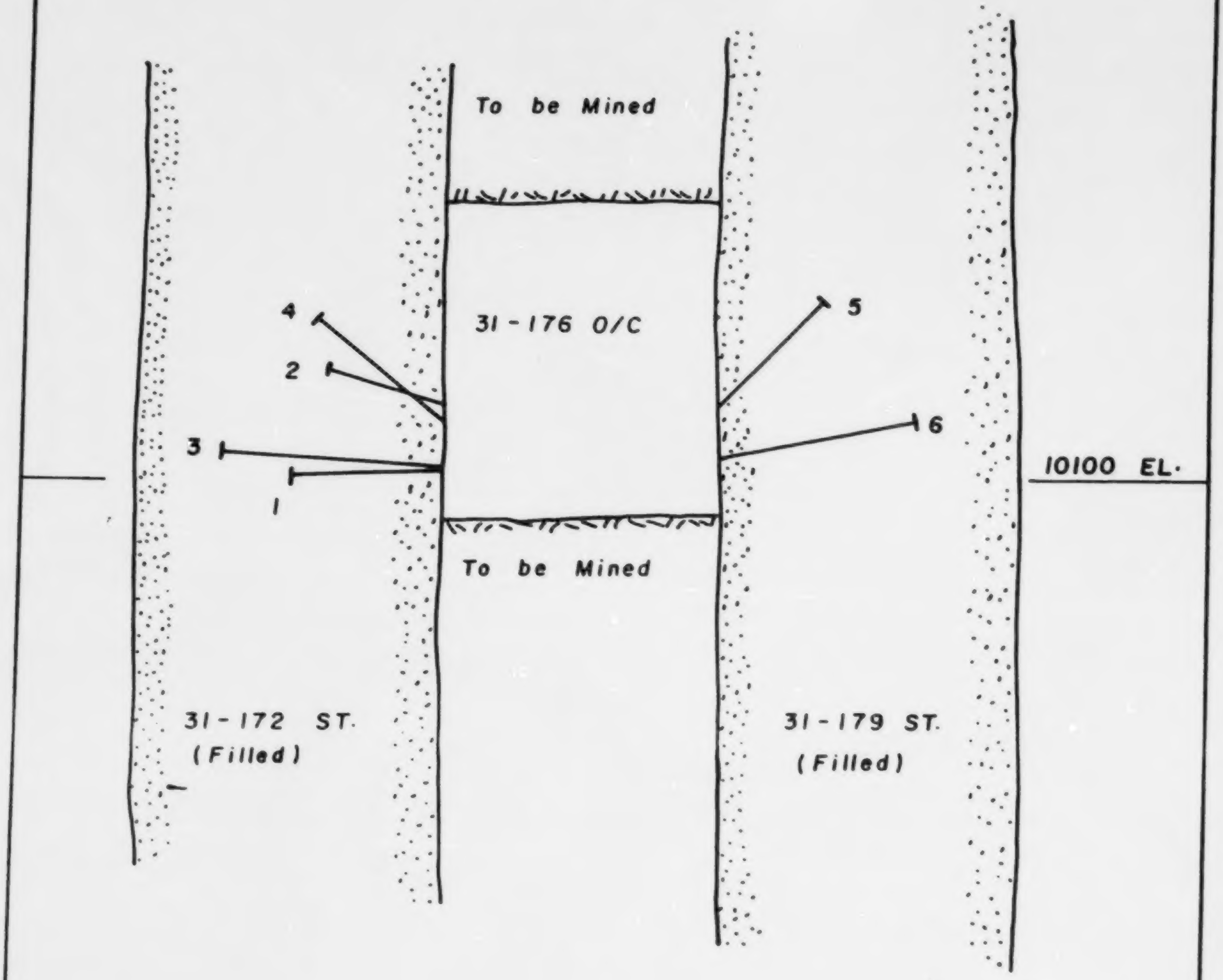
Figure 6


17500 E
17600 E
11900 N
1
2
3
4
5
6
31 - 172 St.
(Filled)
31 - 176 St
Overcut
31 - 179 St.
(Filled)

Drill Hole Locations

Plan View

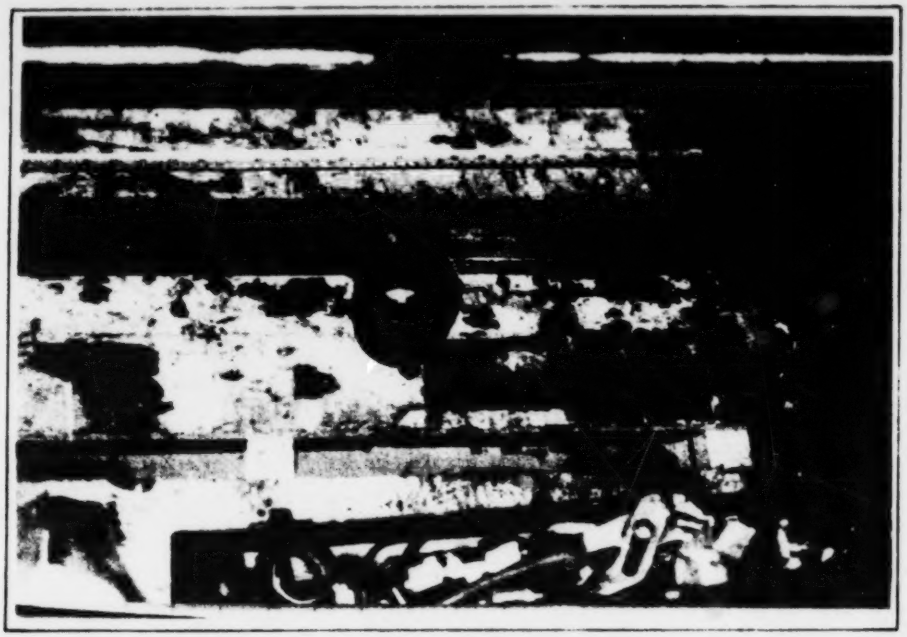
FIG : 7


To be Mined
31-176 O/C
4
2
3
1
5
6
10100 EL.
To be Mined
31-172 ST.
(Filled)
31-179 ST.
(Filled)

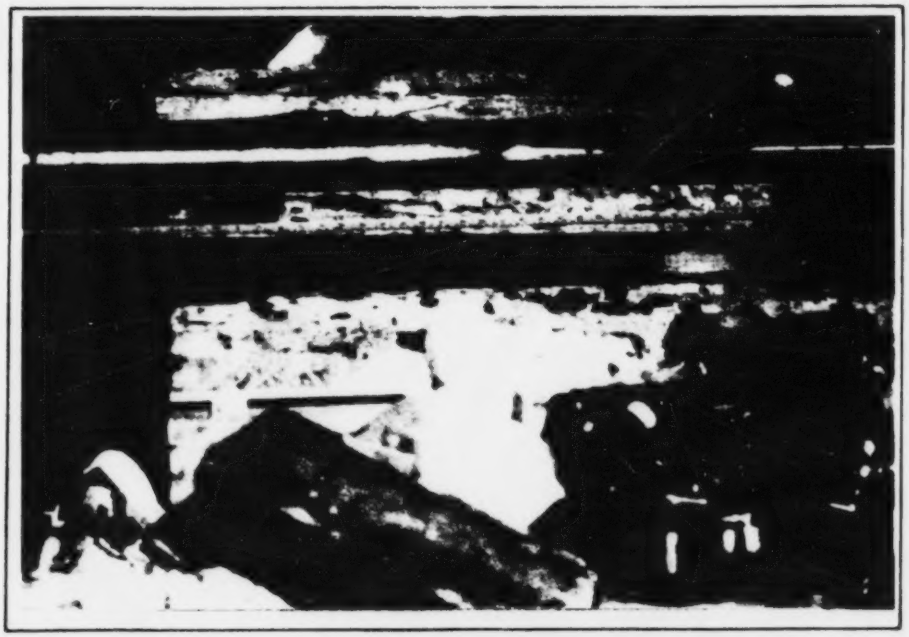
Drill Hole Location

Section View

FIG: 8

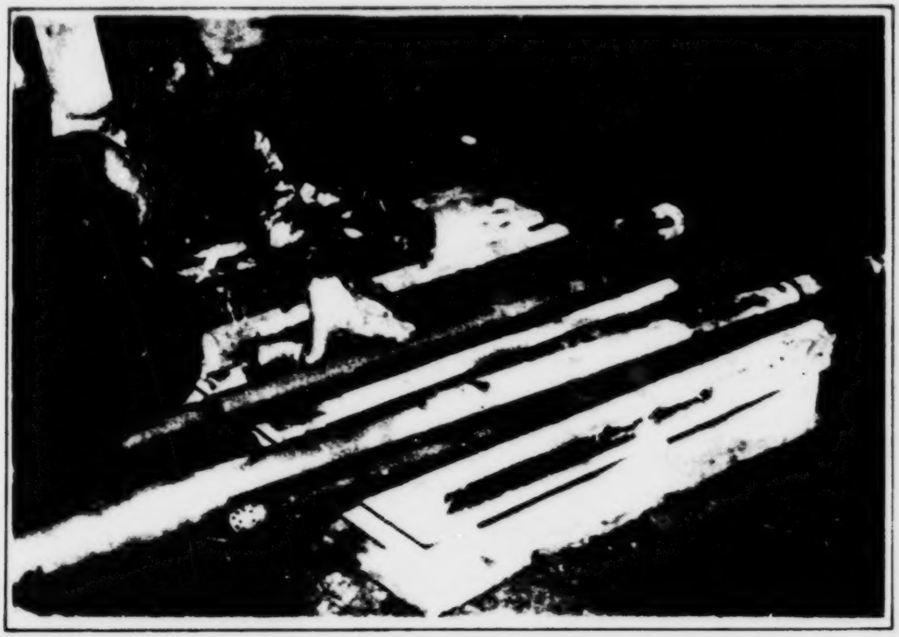


Typical Core Recovery Using
Craelius Split Inner Tube Core Barrel



Core Recovery With Craelius Core Barrel

Split Inner Tube and Standard Core Barrels Figure 9

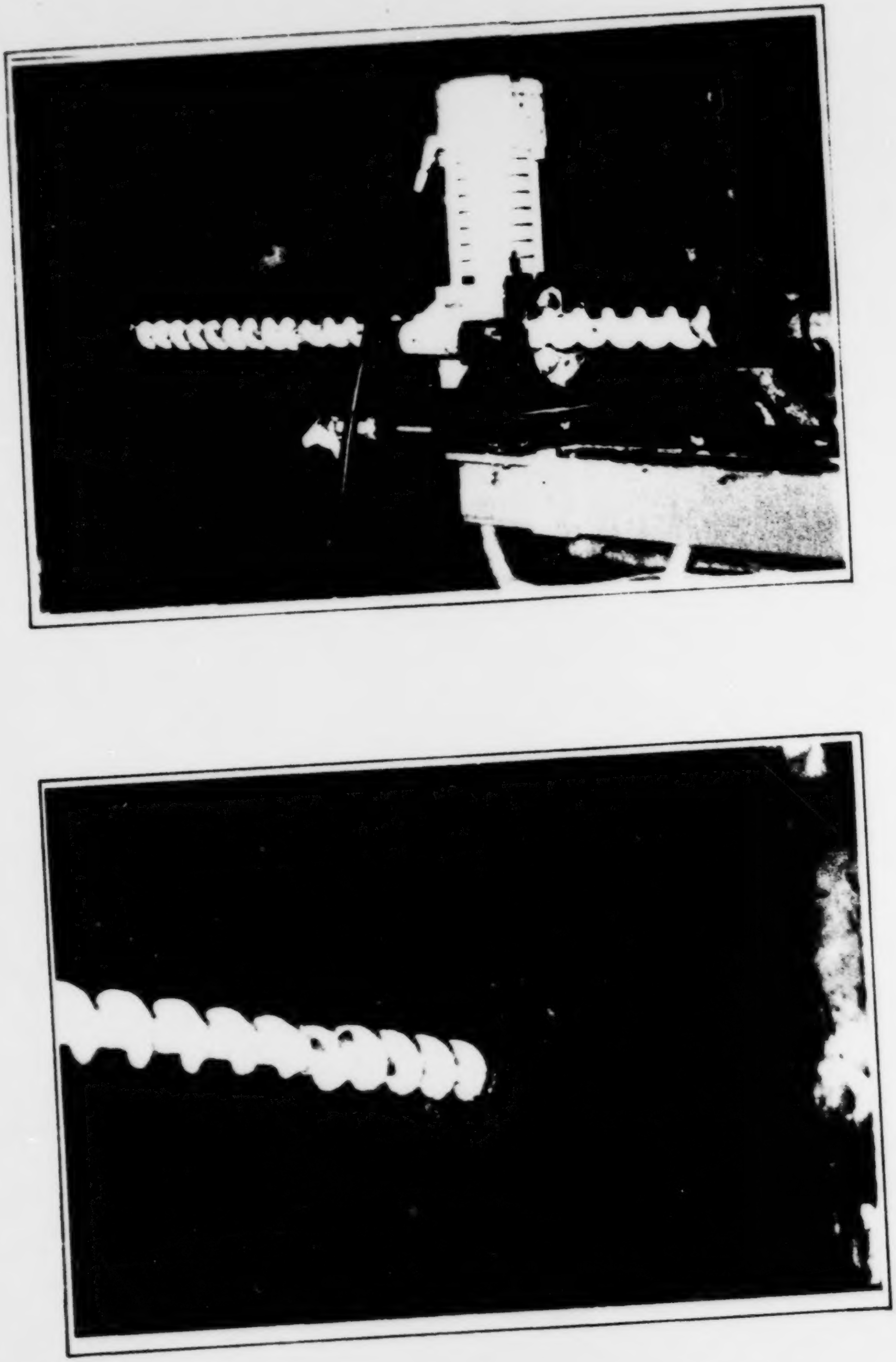


Core Recovery Using NX Solid Inner Tube Core Barrel Samples Are Displayed in a Sample Tray

Example of Core Recovery

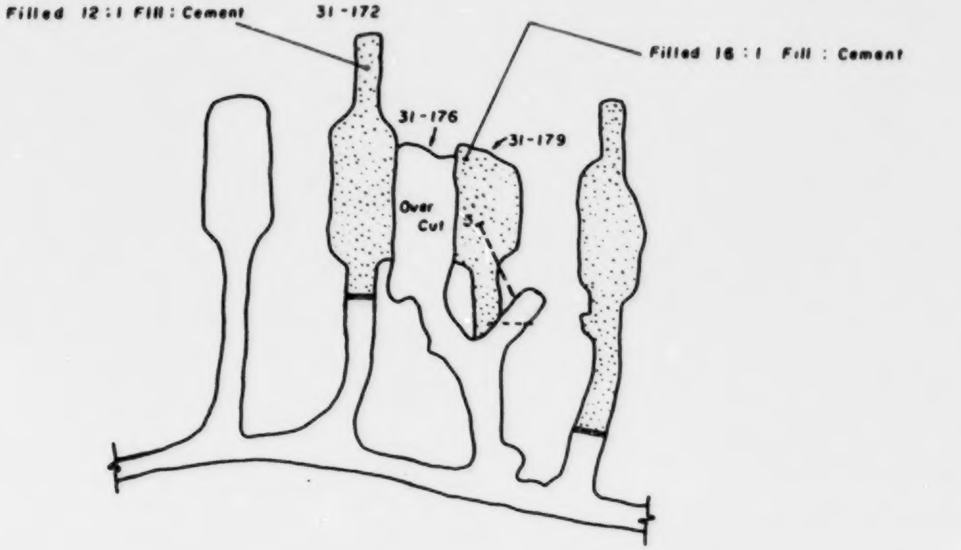
In Solid Inner Tube Core Barrel

Figure 10



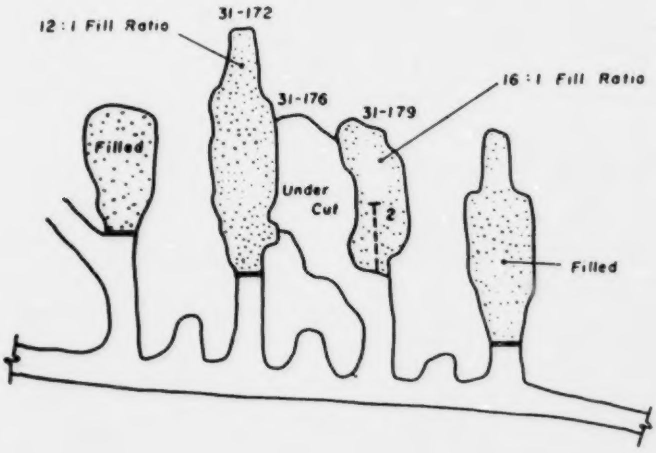
Auger Drilling in Mine Backfill

Figure 11


Filled 12 : 1 Fill : Cement
31 - 172
Filled 16 : 1 Fill : Cement
31 - 176
31 - 179
Over Cut
3

**Note**: Two Glotzl Cells Are Installed in Each Hole

Plan of 29 - 1 Sub - Level

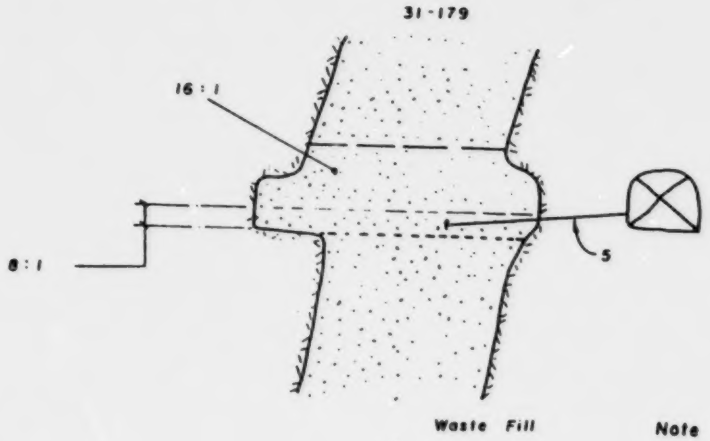

12 : 1 Fill Ratio
31 - 172
31 - 176
31 - 179
16 : 1 Fill Ratio
Filled
Under Cut
2
Filled

Plan of 3125 Level

Glotzl Cell Installation

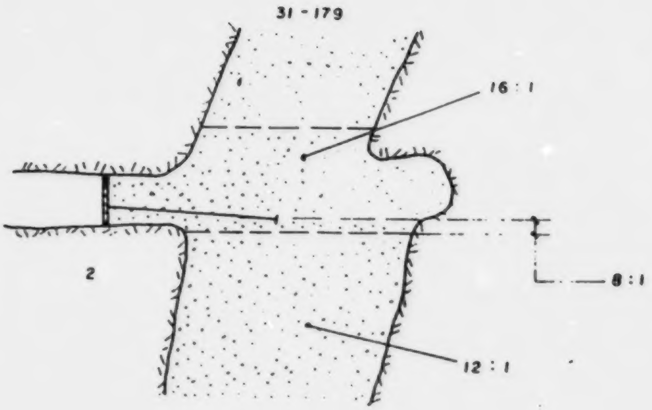
Plan View

FIG : 12


31 - 179
16 : 1
8 : 1
5
Waste Fill

Note : Two Glotzl Cells Are Installed in Each Hole

Section

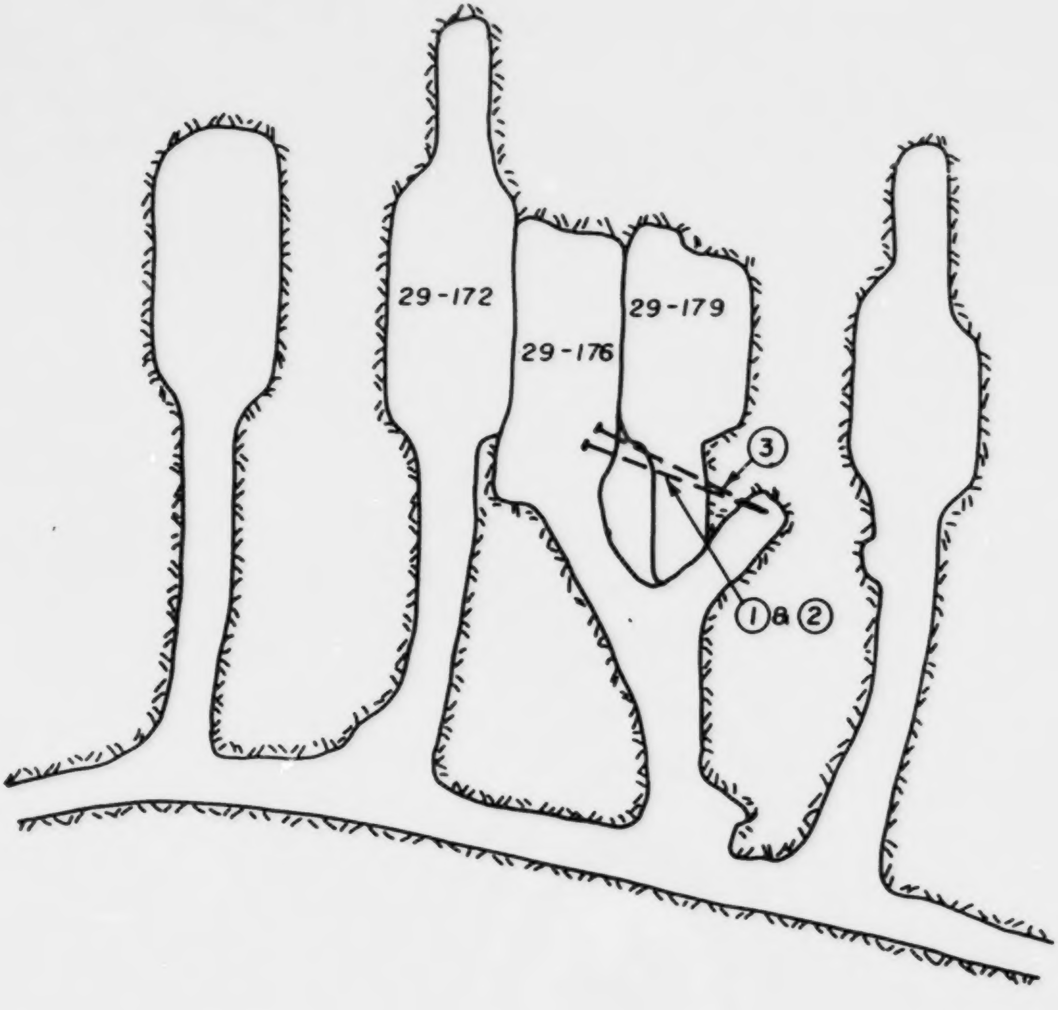

31 - 179
16 : 1
2
8 : 1
12 : 1

Section

Glotzl Cell Installation

Section View

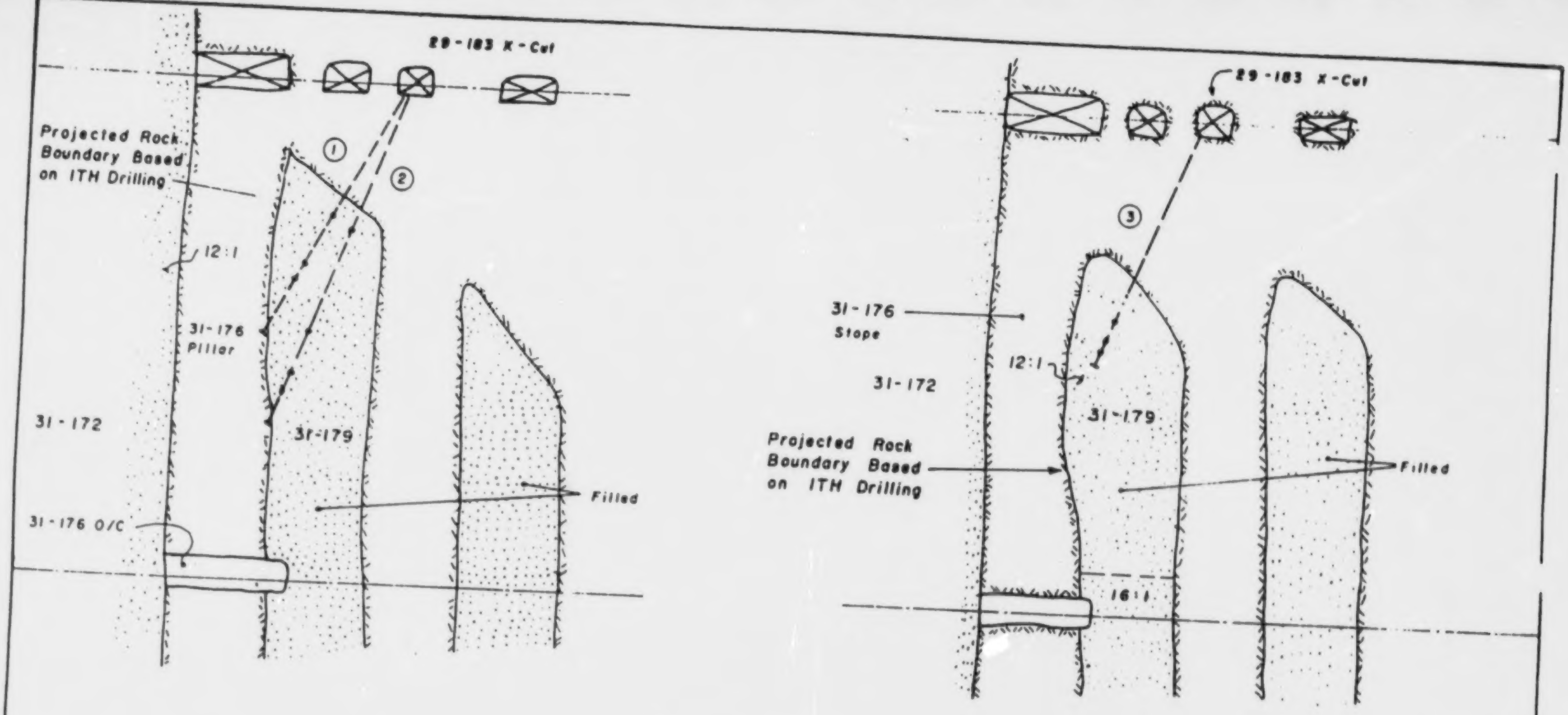
FIG : 13


29-172
29-176
29-179
③
① & ②

Extensometer Installations

Plan View

FIG: 14


29 - 183 X - Cut
Projected Rock Boundary Based on ITH Drilling
①
②
12 : 1
31 - 176 Pillar
31 - 172
31 - 179
Filled
31 - 176 O/C
29 - 183 X - Cut
③
31 - 176 Stope
12 : 1
31 - 172
31 - 179
Projected Rock Boundary Based on ITH Drilling
Filled
16 : 1

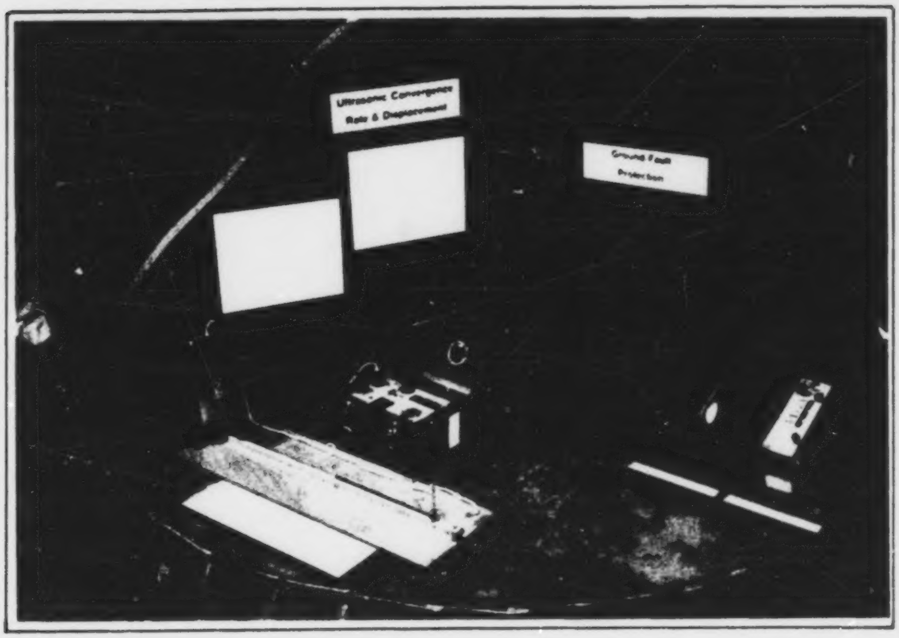
| Hole No. | Length |  |  | Dip | Az. | Hole O |
|---|---|---|---|---|---|---|
|  | Rock | Fill | Total |  |  |  |
| E1 | 48 | 46 | 94 | - 62° | 289° | TGS - 86mm |
| E2 | 47 | 93 | 140 | - 71° | 289° | NX - 75·6mm |
| E3 | 76 | 24 | 100 | - 68° | 280·5° | TGS - 86 mm |

| Hole No. | Anchor Depths |  |  |  |
|---|---|---|---|---|
|  | 4 | 3 | 2 | 1 |
| E1 | 16·7 | 23·7 | 25·9 | - |
| E2 | 17·3 | 31·1 | 36·8 | 39·0 |
| E3 | 24·4 | 27·4 | 28·9 | - |

FIG : 15

Extensometer Installations — Section View

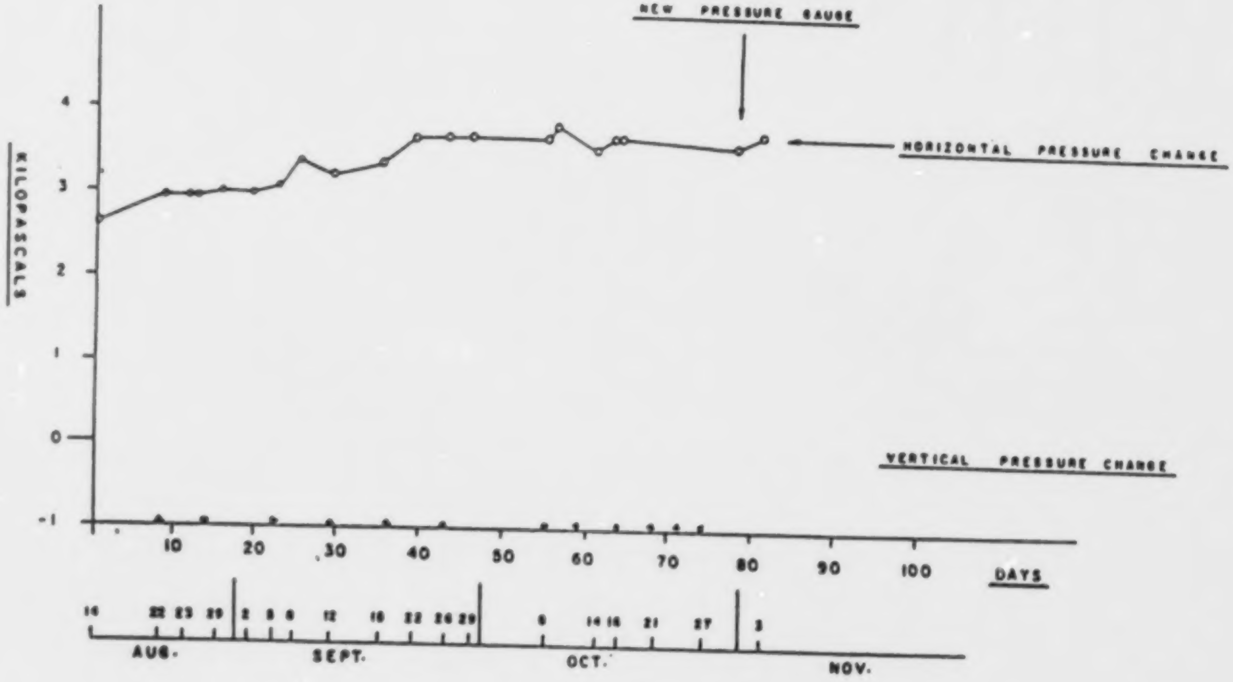
FIG : 15


Ultrasonic Convergence
Rate & Displacement
Ground Fault
Protection

(USBM) United States Bureau of Mines

Ultrasonic Measuring Device

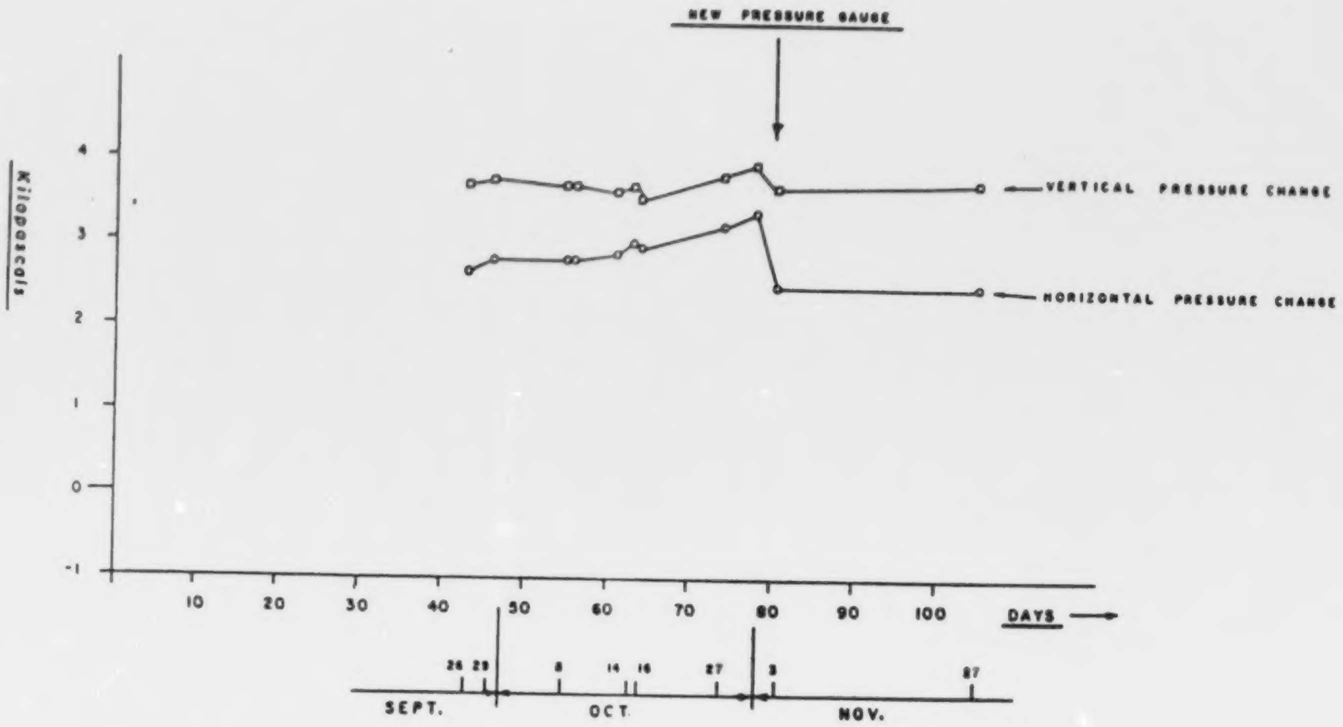
FIG : 16



LOCKERBY MINE

LEVEL PRESSURE CELL — LOCATION 33-179 in 16:1 FILL

FIG: 17

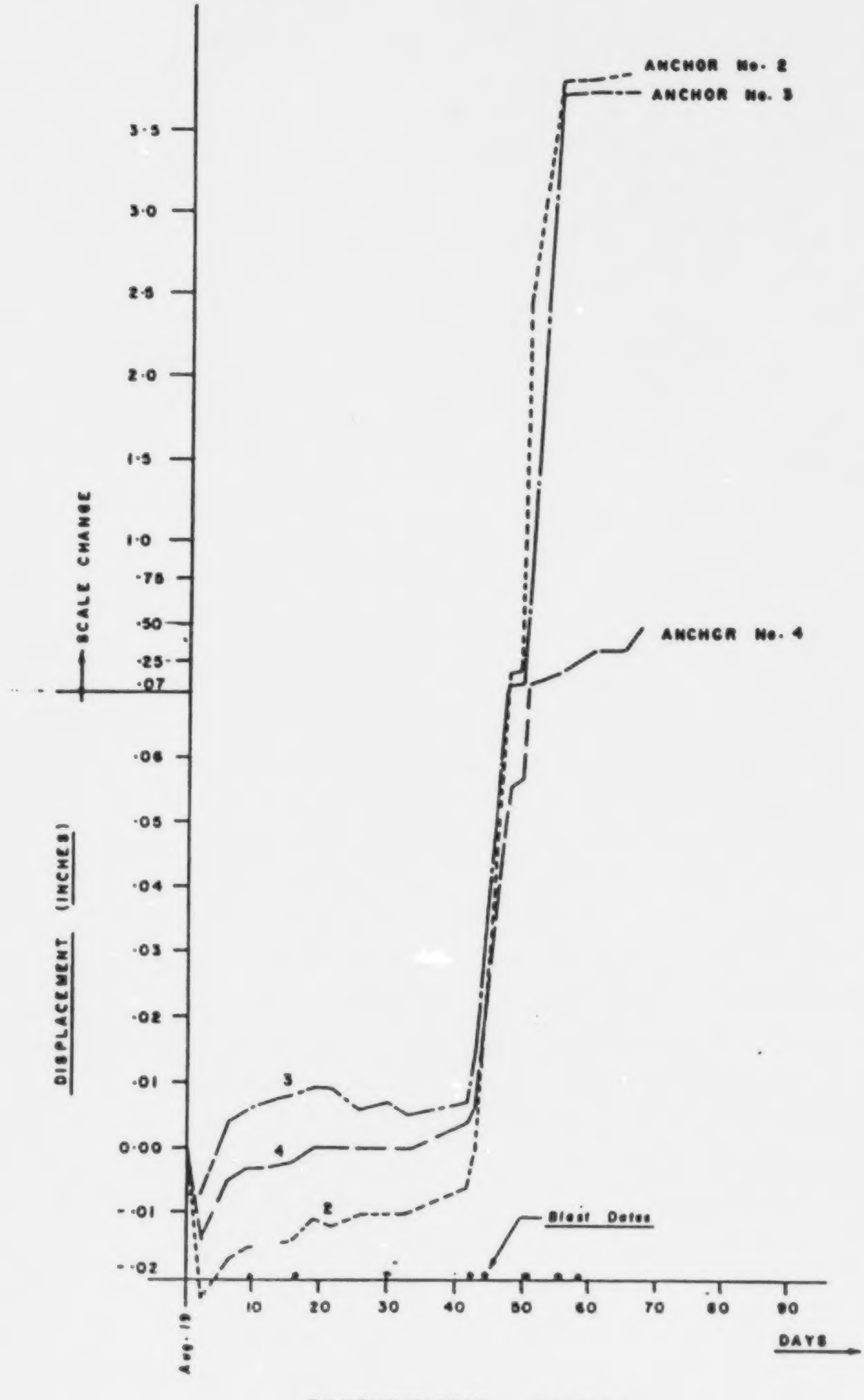


LOCKERBY MINE

29-1 Sub LEVEL PRESSURE CELL — LOCATION 31-179 in 16:1 FILL

FIG : 18

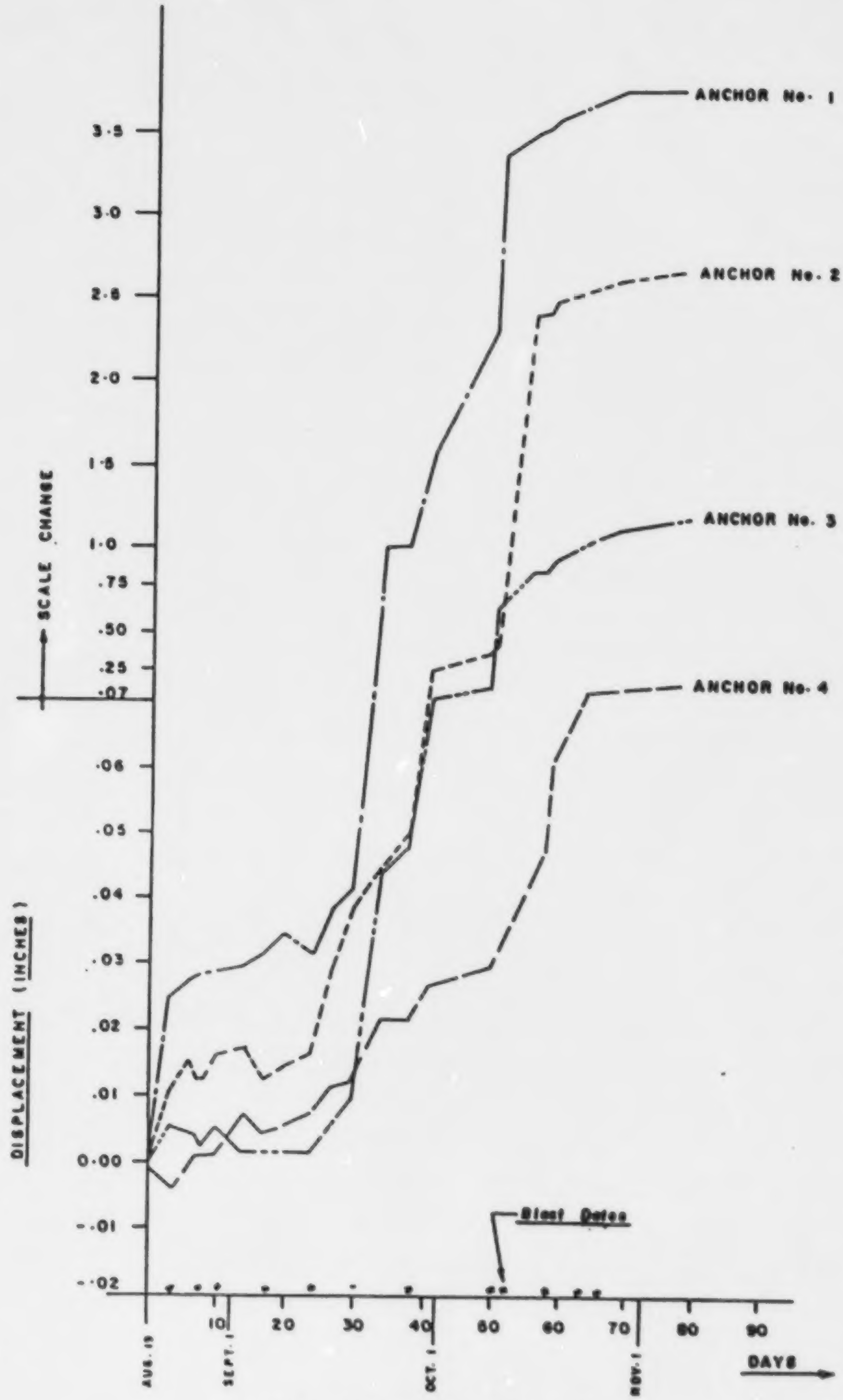
FIG : 18



EXTENSOMETER READING

LOCKERBY MINE — 2900 LEVEL — HOLE E - 1

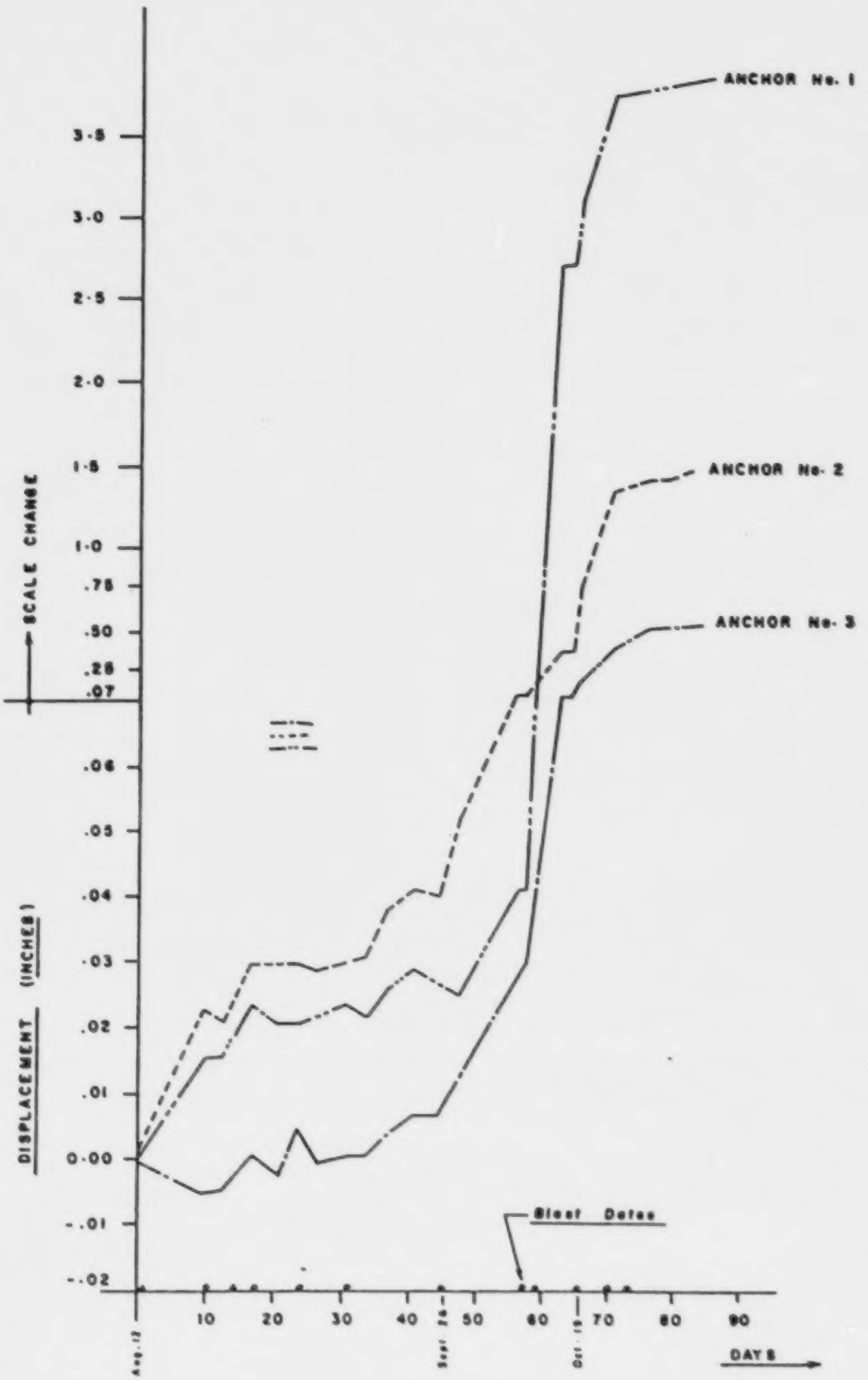
FIG: 19



EXTENSOMETER READING

LOCKERBY MINE — 2900 LEVEL — HOLE E-2

FIG: 20



EXTENSOMETER READING

LOCKERBY MINE — 2900 LEVEL — HOLE E-3

FIG: 21

| | | | |
|---|---|---|---|
| FALCONBRIDGE MINE BACKFILL<br>CANMET SILL MAT PROJECT | DIP 0°<br>AZIMUTH 352° | LOGGED BY: A. BRADFIELD<br>SEPT. 23, 1986 | BOREHOLE N<br>G3 |
| LOCKERBY MINE<br>3125 LEVEL | DRILLED BY CRAELIUS<br>DIAMEC 251/260 | | SHEET<br>1 OF 1 |

| SAMPLE NO. | FOOTAGE (FT.) | DESCRIPTION | REC. (%) | COMP. STR. (psi) | TAILS: CEMENT | MOIS (%) |
|---|---|---|---|---|---|---|
| | | Grey, clay size, high moisture & cement, med. strength all loose | 5 | | | |
| 258 | 5 | Grey, silt size, high strength, med. cement & moisture no loose | 100 | 130:3 | 17:1 | 16. |
| | 10 | Light grey, clay size, high strength & cement, low moisture, all loose | | | | |
| 257 | | Very light grey, clay to silt size, very high strength & cement, low moisture, no loose | 95 | 521.1 | 8:1 | 12. |
| | 15 | Very light grey, clay size, very high strength & cement, low moisture, 10% loose | 95 | | | |
| | 20 | Very light grey, clay to silt size, very high strength & cement, low moisture, no loose | 98 | | | |
| 259 | | | | 448.7 | 11:1 | 12. |
| | 25 | Very lt. gr. ,cl. to silt size, very high str. cem., low moist., 10% loose. At approx. 26.5ft, drk. med. moist., sand size for 6". | 90 | | | |
| | END30 | | | | | |
| | | OVERALL RECOVERY | 81 | | | |

REMARKS: PERSONAL CONVENTIONS USED

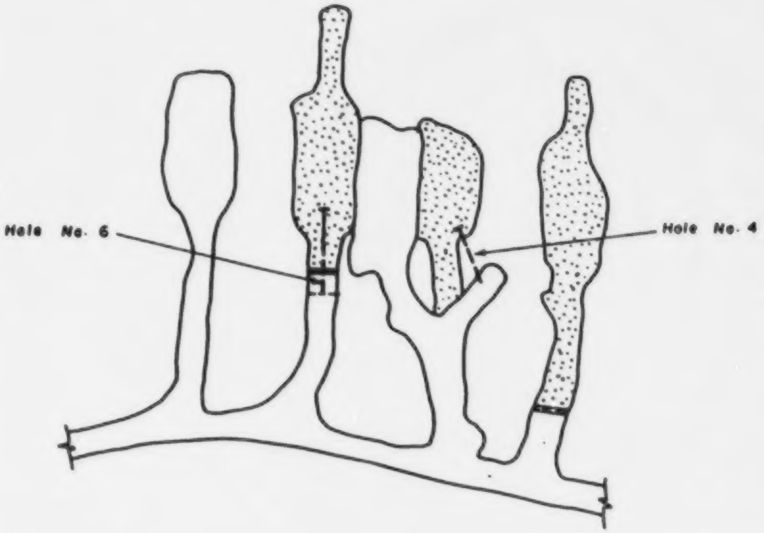
AVG.RANGES FOR COMP.STR. LOW 0 - 50 psi
MED. 50 - 100 psi
HIGH > 100 psi

SIZES: Low tails, high cement = clay size
Med tails, med. cement = silt size
High tails, low cement = sand size

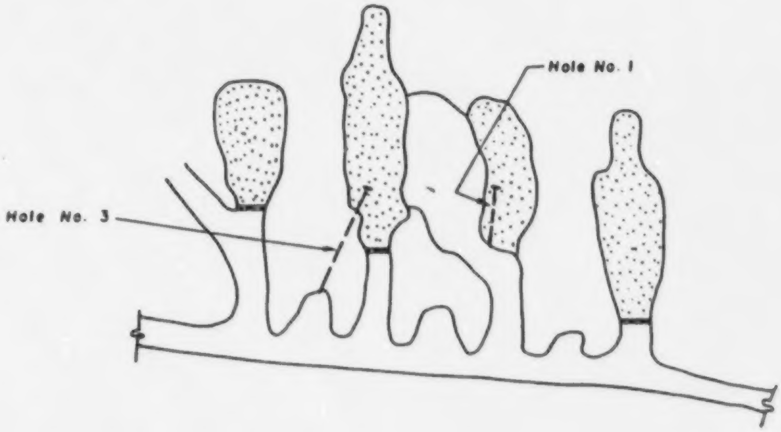
86-248

*Drill Core Mapping — Example*

*FIG : 22*


Hole No. 6
Hole No. 4

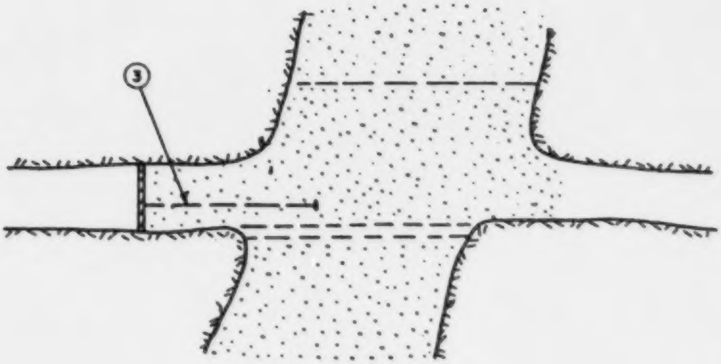
Plan of 29-1 Sub-Level


Hole No. 1
Hole No. 3

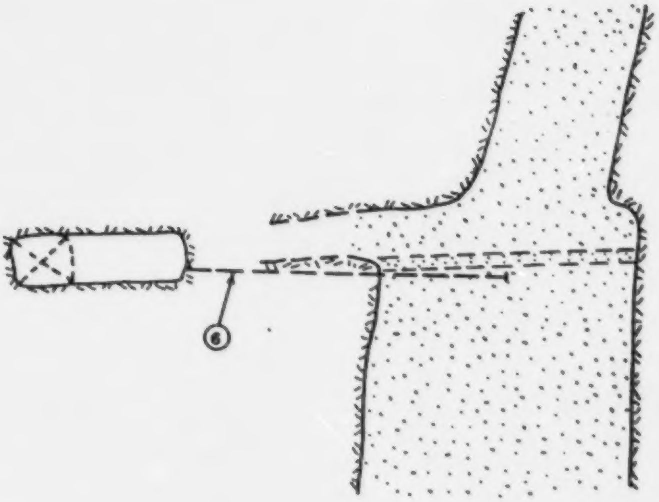
Plan of 3125 Level

Geophone Installations — Plan View

FIG: 23



Section



Section

Geophone Installations — Section View

FIG: 24

# COMPARISON OF ROCK to FILL

VIBRATION LEVELS AT EQUIVALENT SCALED DISTANCES

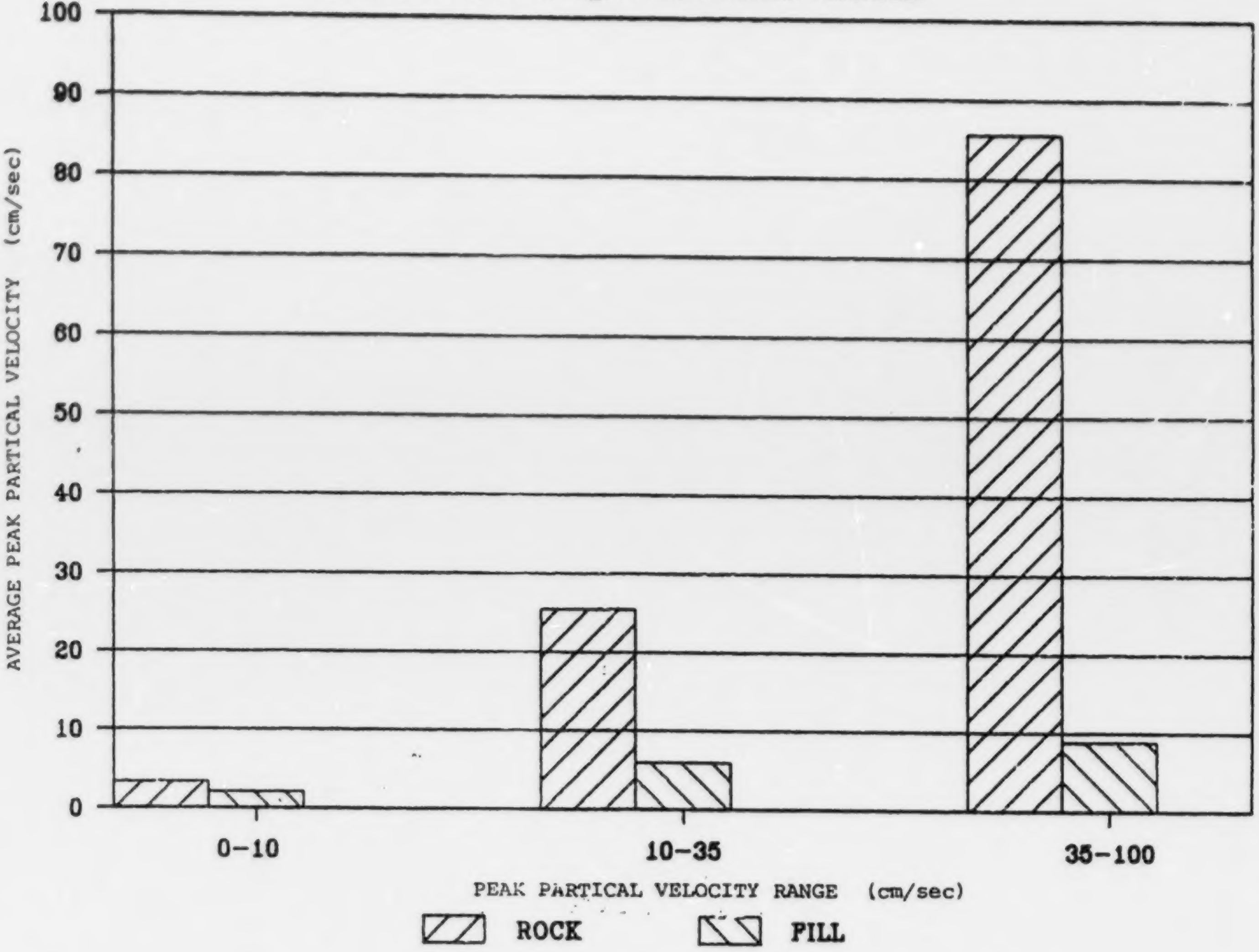


FIG: 25

# VIBRATION ATTENUATIONS--- ROCK vs FILL

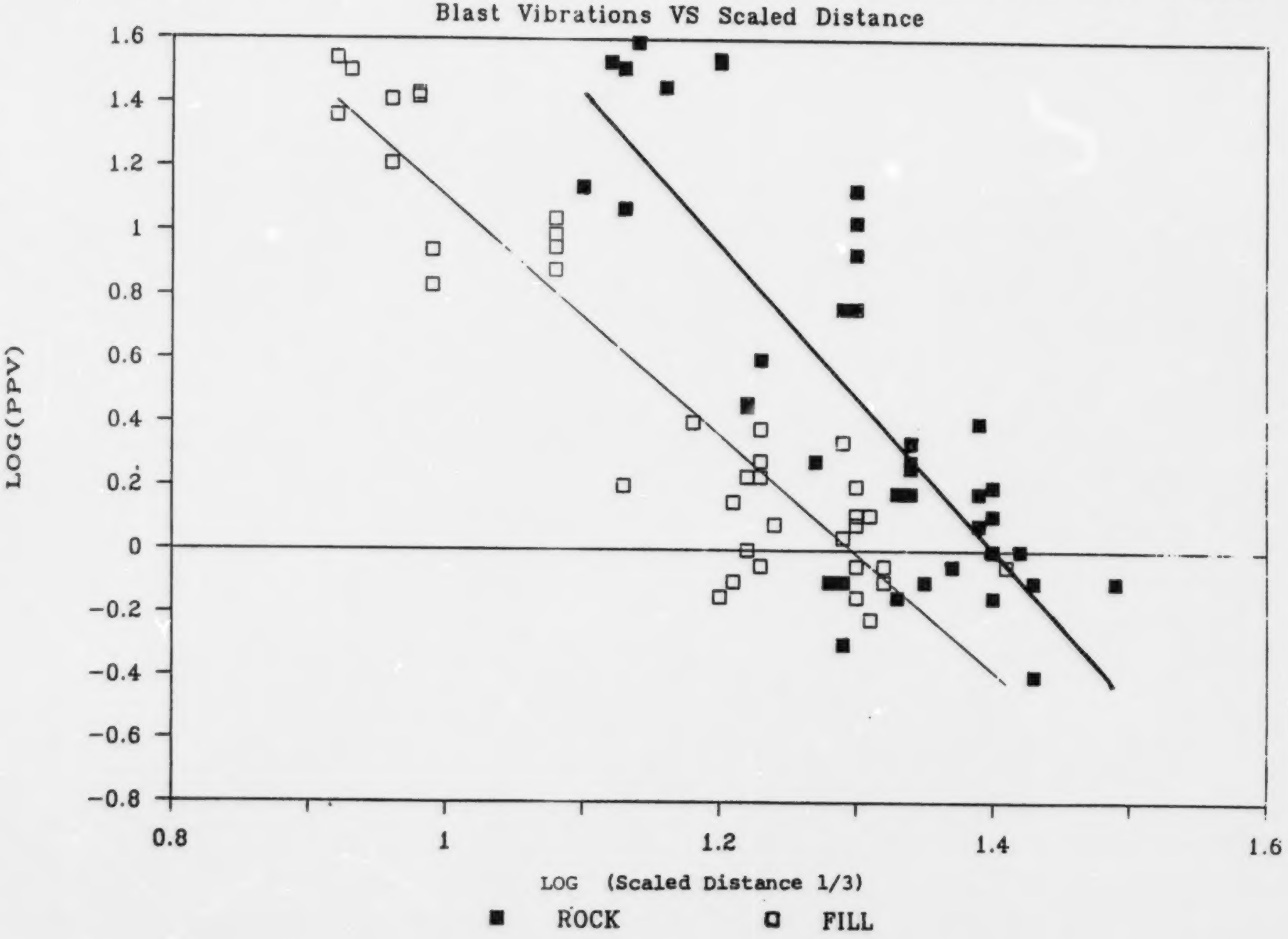


FIG: 26

# CEMENT CONTENT ***22:1 vs 12:1 ***

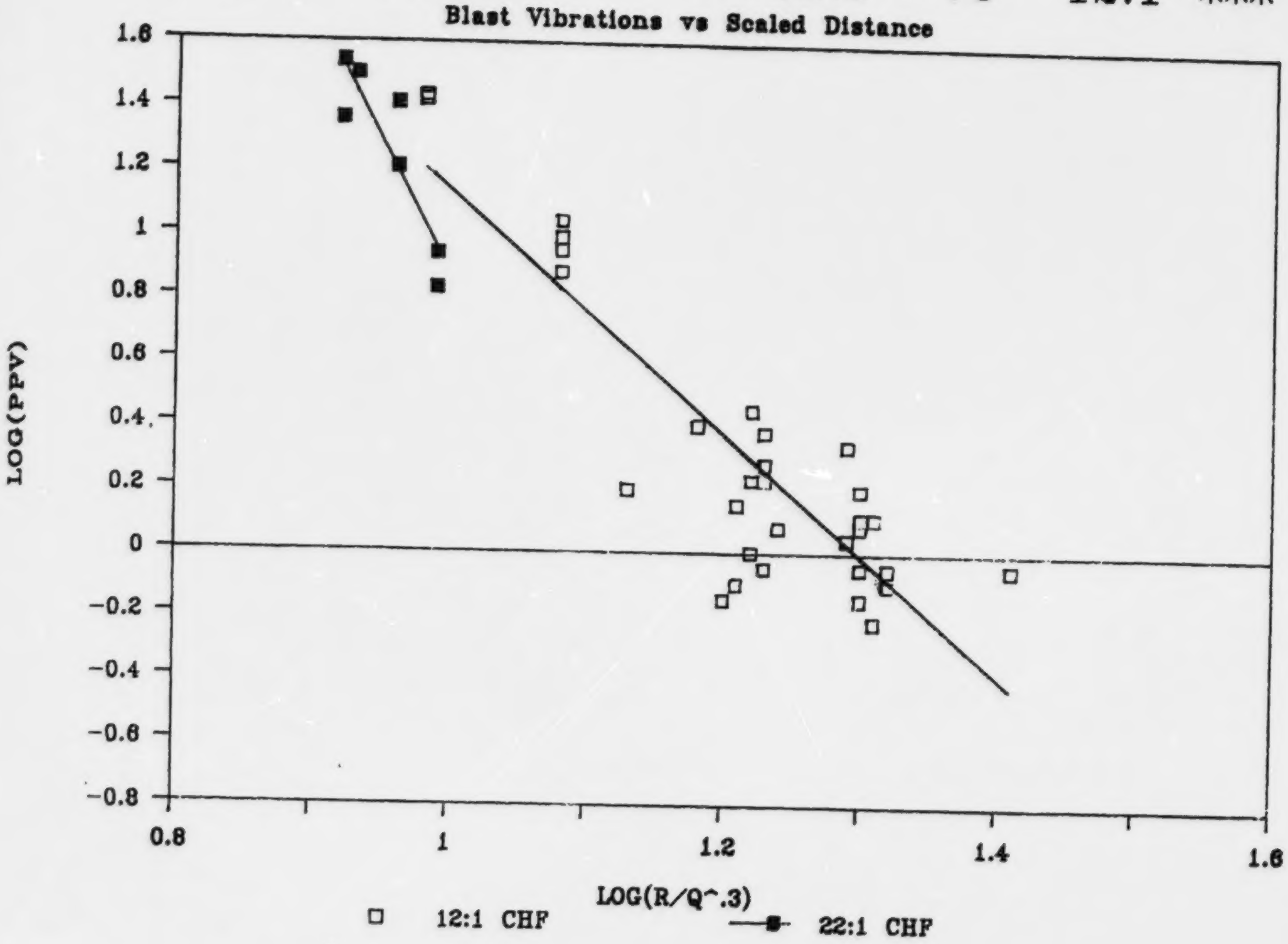
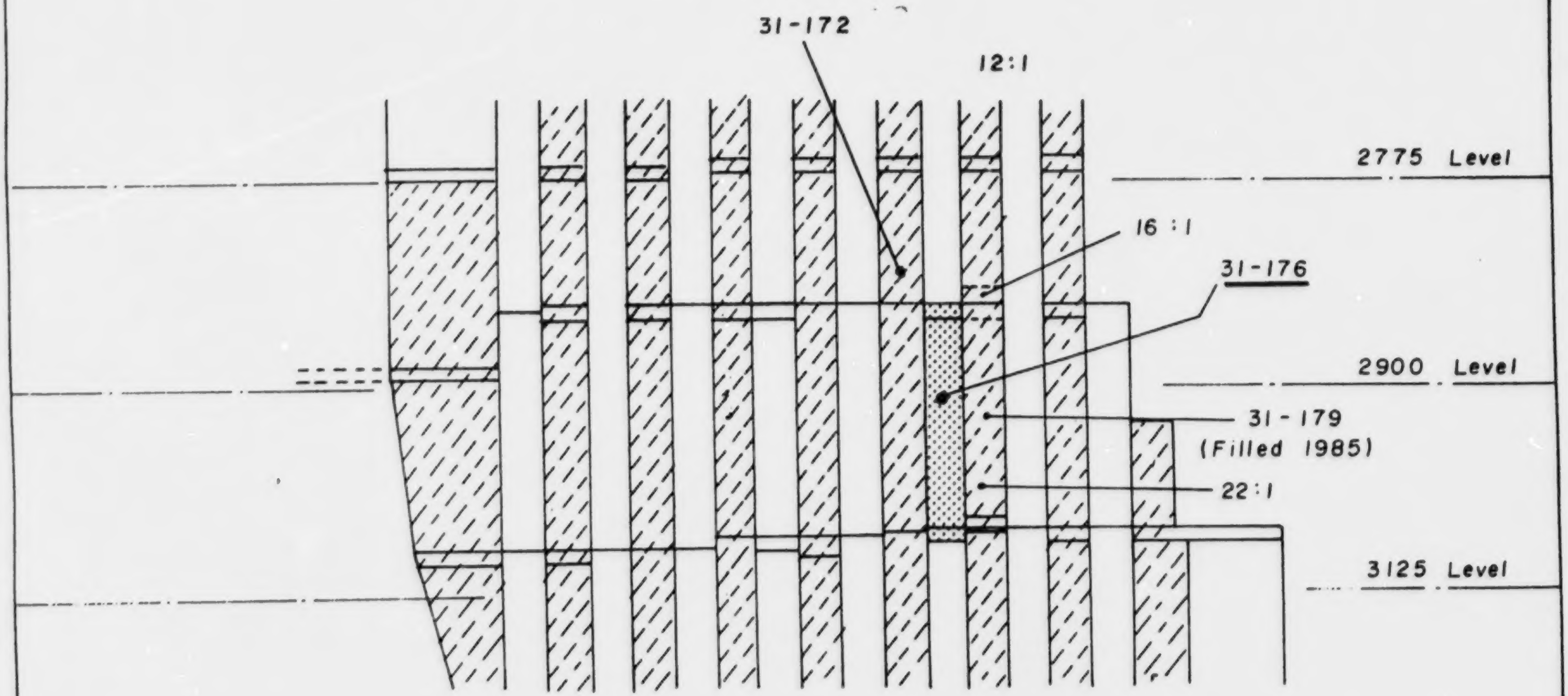


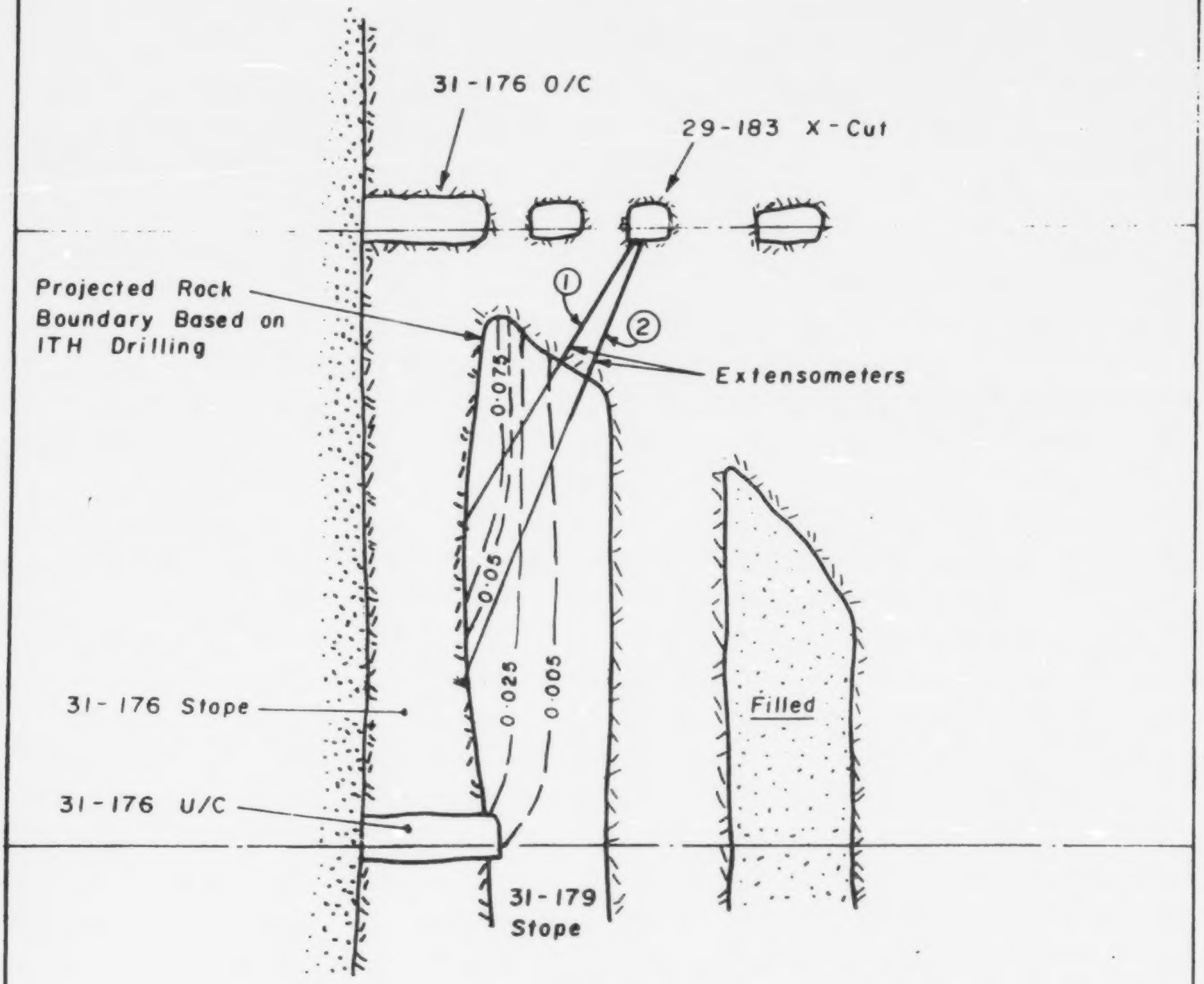
FIG : 27


31-172
12:1
2775 Level
16 :1
31-176
2900 Level
31 - 179
(Filled 1985)
22:1
3125 Level

31 - 176 Stope — Longitudinal Section — Stope Location

FIG : 26

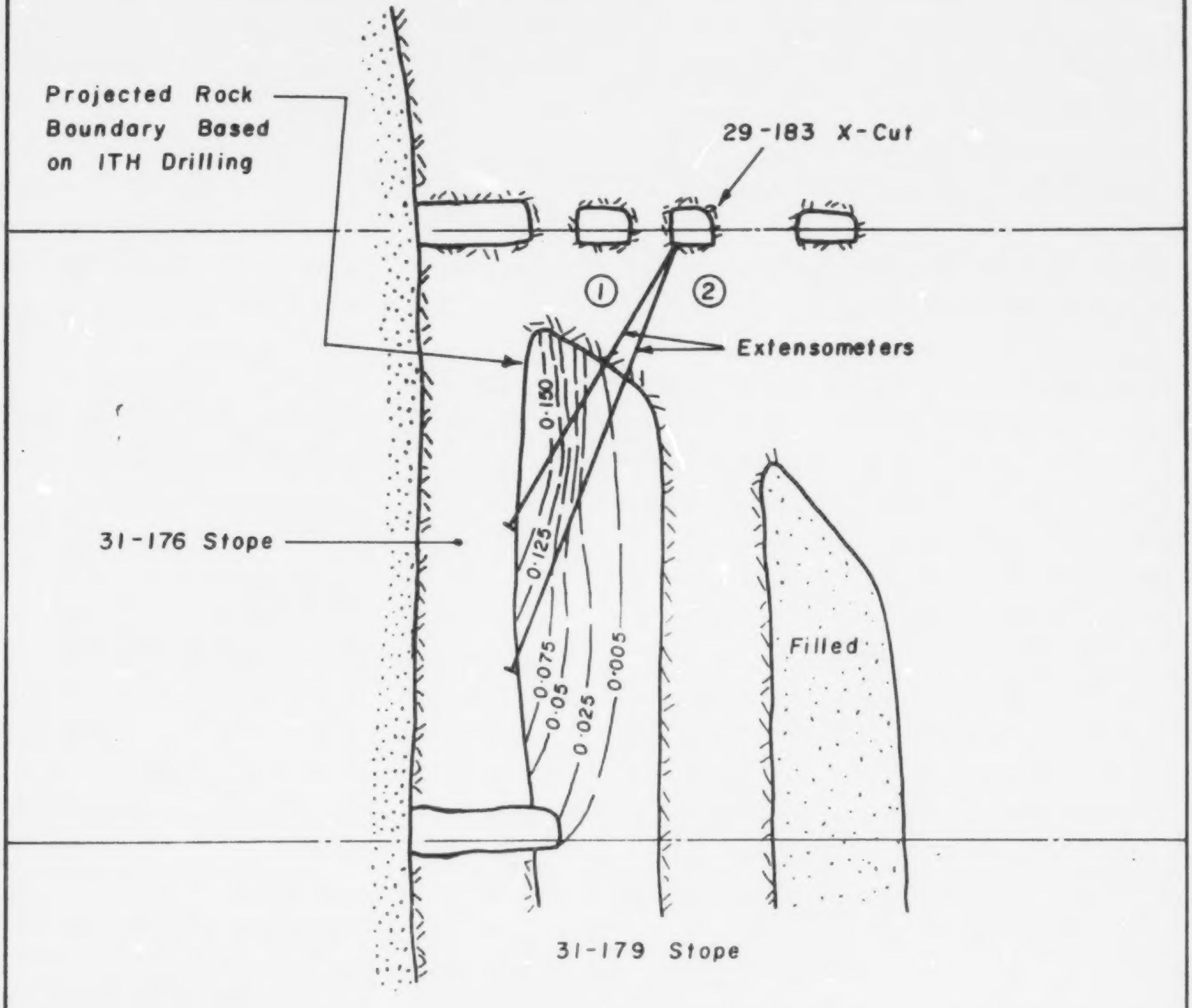
FIG : 28


31-176 O/C
29-183 X-Cut
Projected Rock Boundary Based on ITH Drilling
①
②
Extensometers
0·075
0·05
0·025
0·005
31-176 Stope
31-176 U/C
31-179 Stope
Filled

Backfill Displacement Contours

X - Direction

FIG : 29


Projected Rock Boundary Based on ITH Drilling
29-183 X-Cut
①
②
Extensometers
0·150
0·125
0·075
0·05
0·025
0·005
31-176 Stope
Filled
31-179 Stope

Back Displacement Contours

Y - Direction

FIG : 30

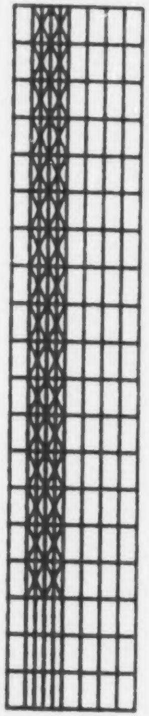
JOB TITLE : CANMET \ FALCONBRIDGE SILL MAT PROJECT

*MUDEC (Version 1.00)*

LEGEND

```
 0/ 0/   0   0: 0
cycle   2500
-3.480E+01 <.x <   4.880E+01
-7.980E+01 < y <   3.798E+00

BLOCK plot
```



Mudec Model Layout

FIG : 31

## FILL MATERIAL PROPERTIES USED AS MUDEC INPUT

| PROPERTY ONE | PROPERTY TWO |
|---|---|
| Bulk Modulus = 50E6 Pa | Bulk Modulus = 50E6 Pa |
| Shear Modulus = 37E6 Pa | Shear Modulus = 37E6 Pa |
| Cohesion = 0.2E6 Pa | Cohesion = 0.0 Pa |
| Friction (Joints) = 35 deg | Friction (Joints) = 20 deg |
| Tensile Str. (Joints) = 1E3 Pa | Tensile Str. (Joints) = 0.0 Pa |
| Density = 2100 kg/m*m*m | Density = 2100 kg/m*m*m |

CONSTITUTIVE RELATION

Blocks = Elastic      Joints = Mohr Coulomb

FILL MATERIAL PROPERTIES

FIG: 32

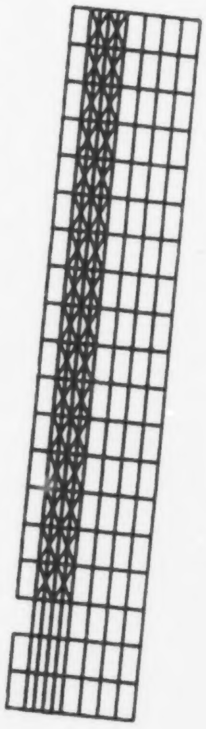
JOB TITLE : CANMET \ FALCONBRIDGE SILL MAT PROJECT

*MUDEC (Version 1.00)*

LEGEND

```
 0/ 0/    0    0: 0
cycle   5000
-3.481E+01 < x <   4.879E+01
-7.980E+01 < y <   3.798E+00

BLOCK plot
```



Mudec Model Layout

Undercut Removed

FIG : 33

JOB TITLE : CANMET \ FALCONBRIDGE SILL MAT PROJECT

*MUDEC (Version 1.00)*

LEGEND

```
 0/ 0/   0   0: 0
cycle   5200
-3.481E+01 < x <   4.879E+01
-7.980E+01 < y <   3.798E+00

BLOCK plot
XX-stress  contours
Contour interval=   2.500E+05
Number of contours/color=   3
Min=-3.000E+06 Max= 0.000E+00
```

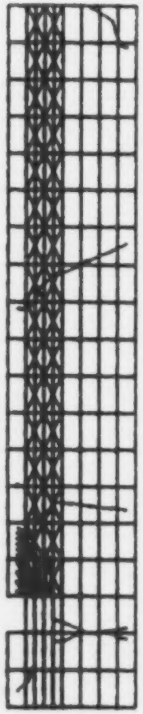


FIG : 34

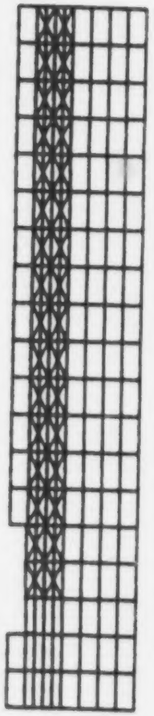
JOB TITLE : CANMET \ FALCONBRIDGE SILL MAT PROJECT

MUDEC (Version 1.00)

LEGEND

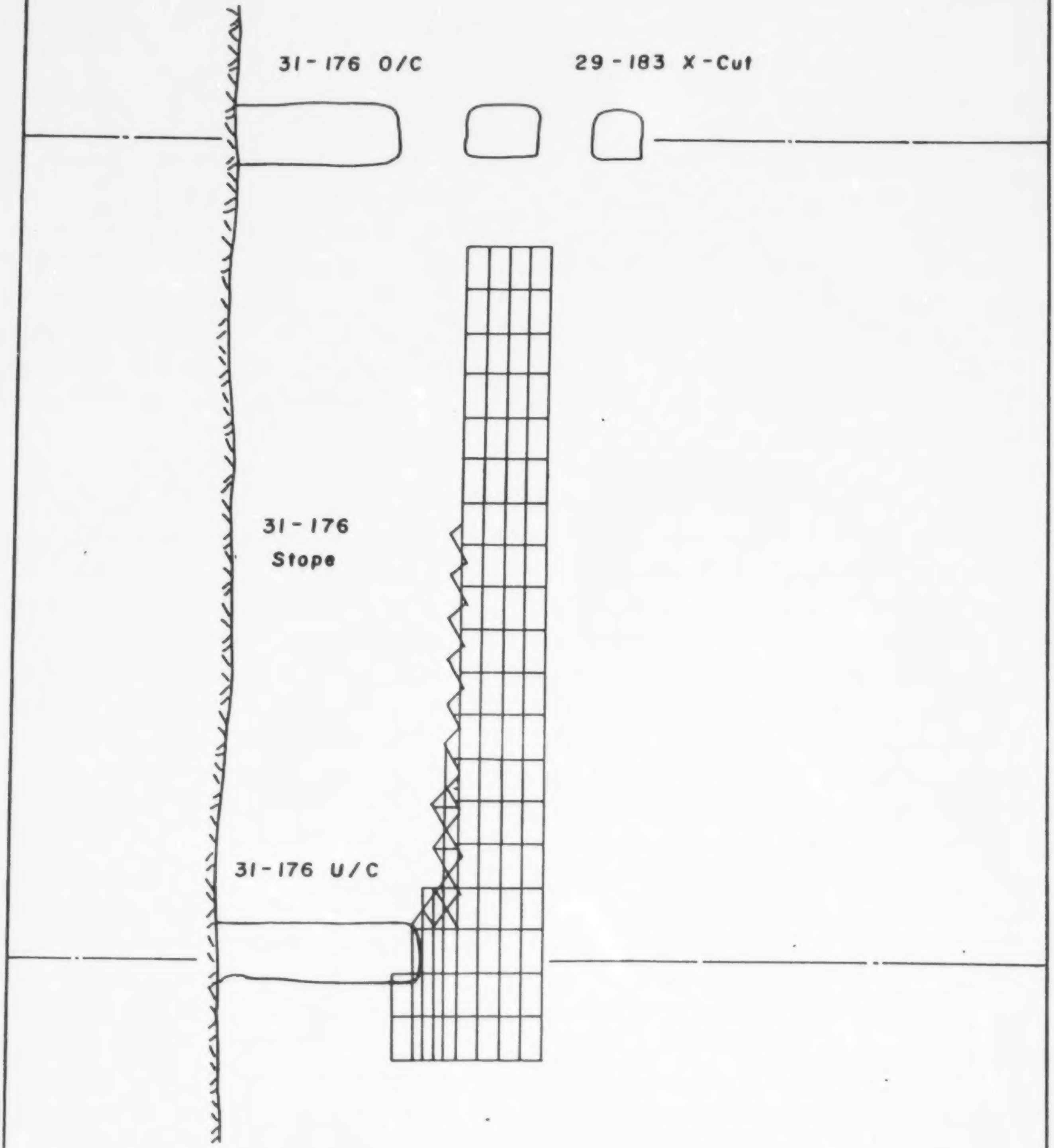
```
 0/ 0/   0   0: 0
cycle  16000
-3.481E+01 < x <   4.879E+01
-7.980E+01 < y <   3.798E+00

BLOCK plot
```



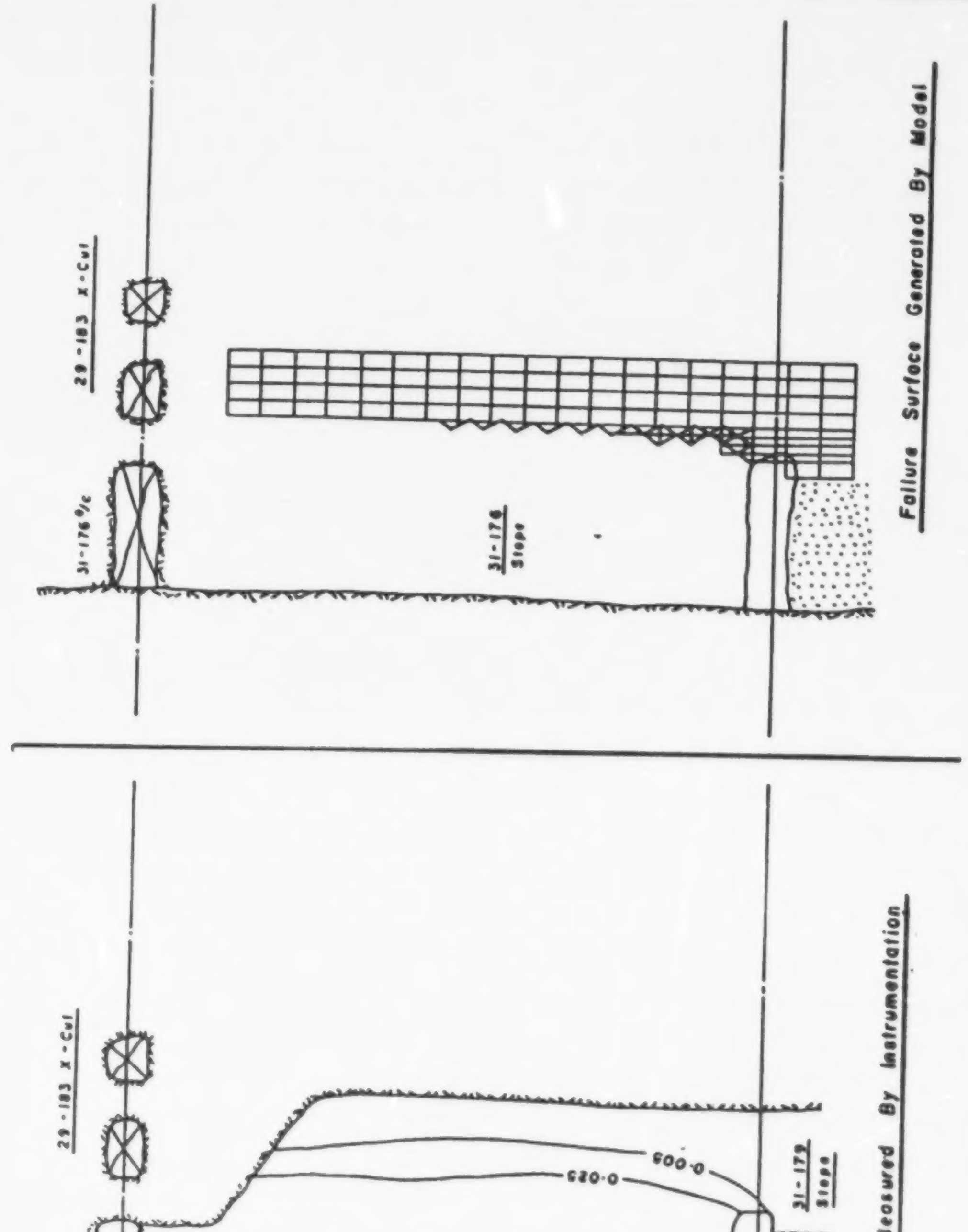
Mudec Model — Mining Cycle

FIG : 35


31-176 O/C
29-183 X-Cut
31-176
Stope
31-176 U/C

Model Final Profile

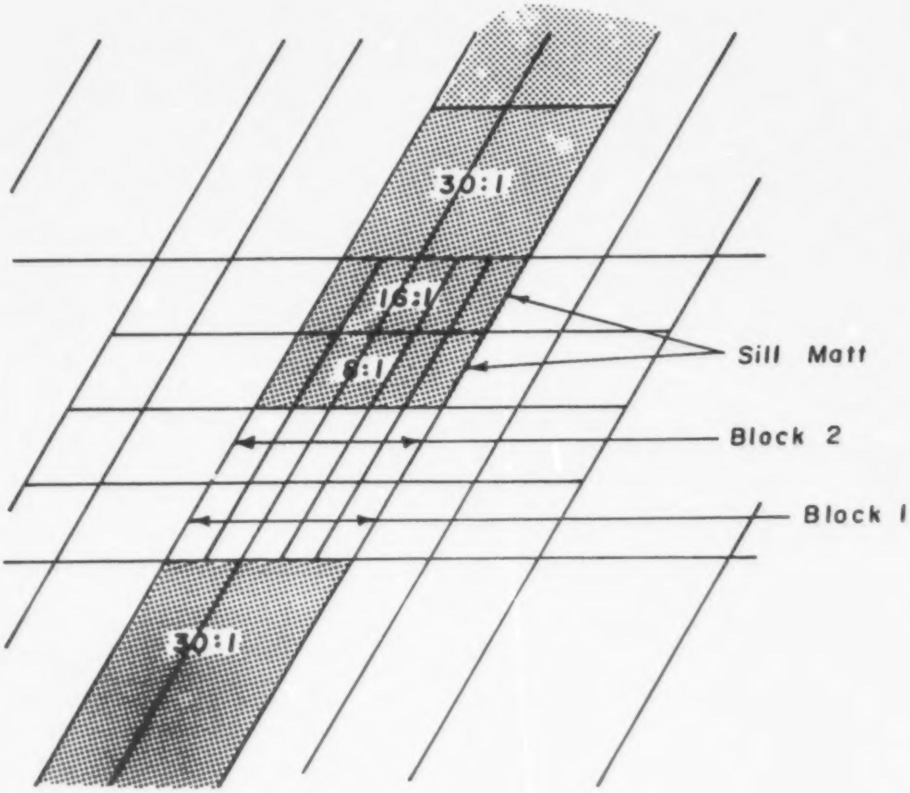
FIG : 36


29-183 X-Cut
31-176 O/C
31-176 Slope
Failure Surface Generated By Model
29-183 X-Cut
31-176 O/C
0.005
0.025
31-176 Slope
31-179 Slope
Failure Surface Measured By Instrumentation

Comparison of Failure Surfaces

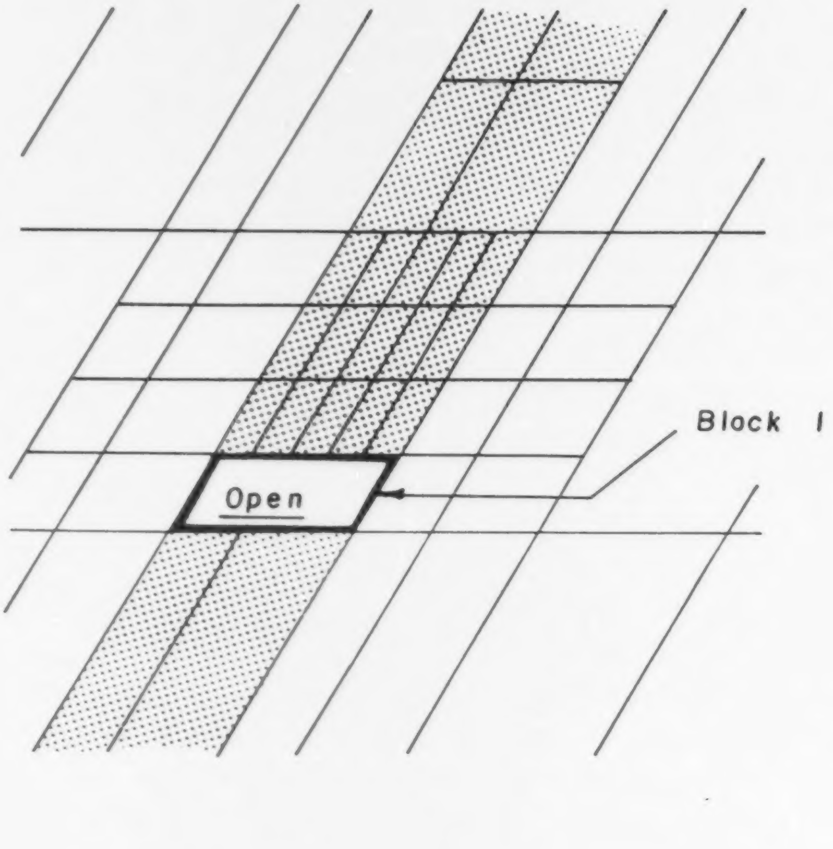
Actual vs Modelled

FIG : 37


30:1
16:1
8:1
Sill Matt
Block 2
Block 1
30:1

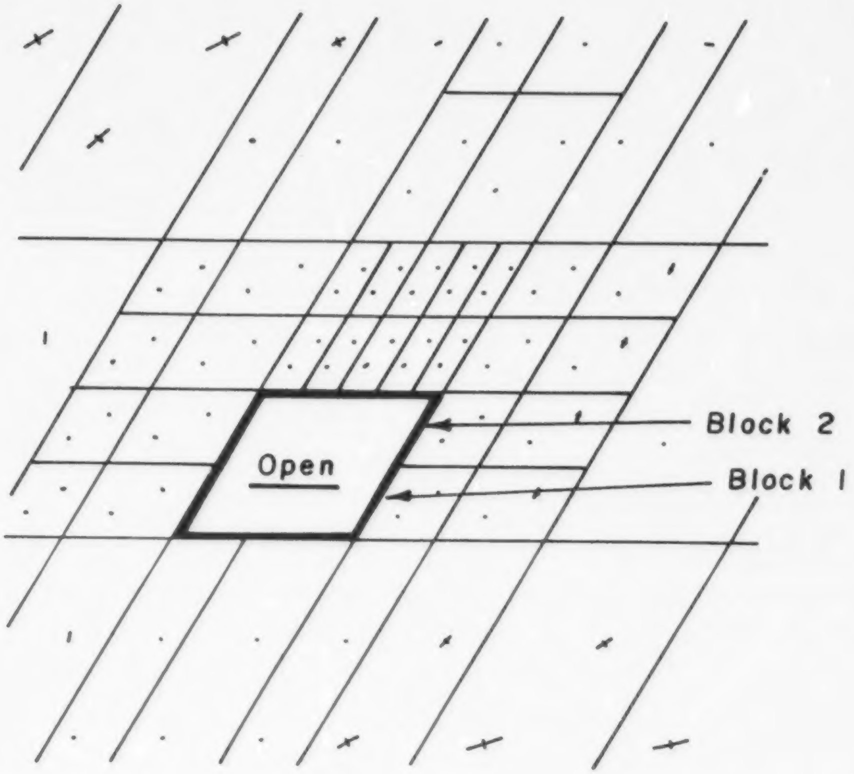
Mining Below a Sill Mat

FIG : 38


Block I
Open

Removal of Second Last Cut

FIG : 39


Open
Block 2
Block 1

Removal of Final Cut

FIG : 40

FALCONBRIDGE

[redacted]

CONTRACT SERIAL NO. OSQ85-00292
CONTRACT REQUISITION NO. 23440-5-9205
CONTRACTOR: FALCONBRIDGE LTD.

# IN SITU MONITORING AND COMPUTER MODELING OF A CEMENTED SILL MAT AND CONFINES DURING TERTIARY STAGE PILLAR RECOVERY

## APPENDICES

A - Lockerby Mine Overview
B - Craelius Drill Data
C - Trow Ontario Limited Report
D - Stope Void Profiles
E - Drill Core Mapping
F - Blast Vibration Monitoring Results
G - State-Of-The-Art Review on Backfilling
Technology.

125 pp

MICROMEDIA

# APPENDIX A

## LOCKERBY MINE OVERVIEW

FALCONBRIDGE LIMITED

SUDBURY OPERATIONS

GUIDE BOOK - LOCKERBY MINE

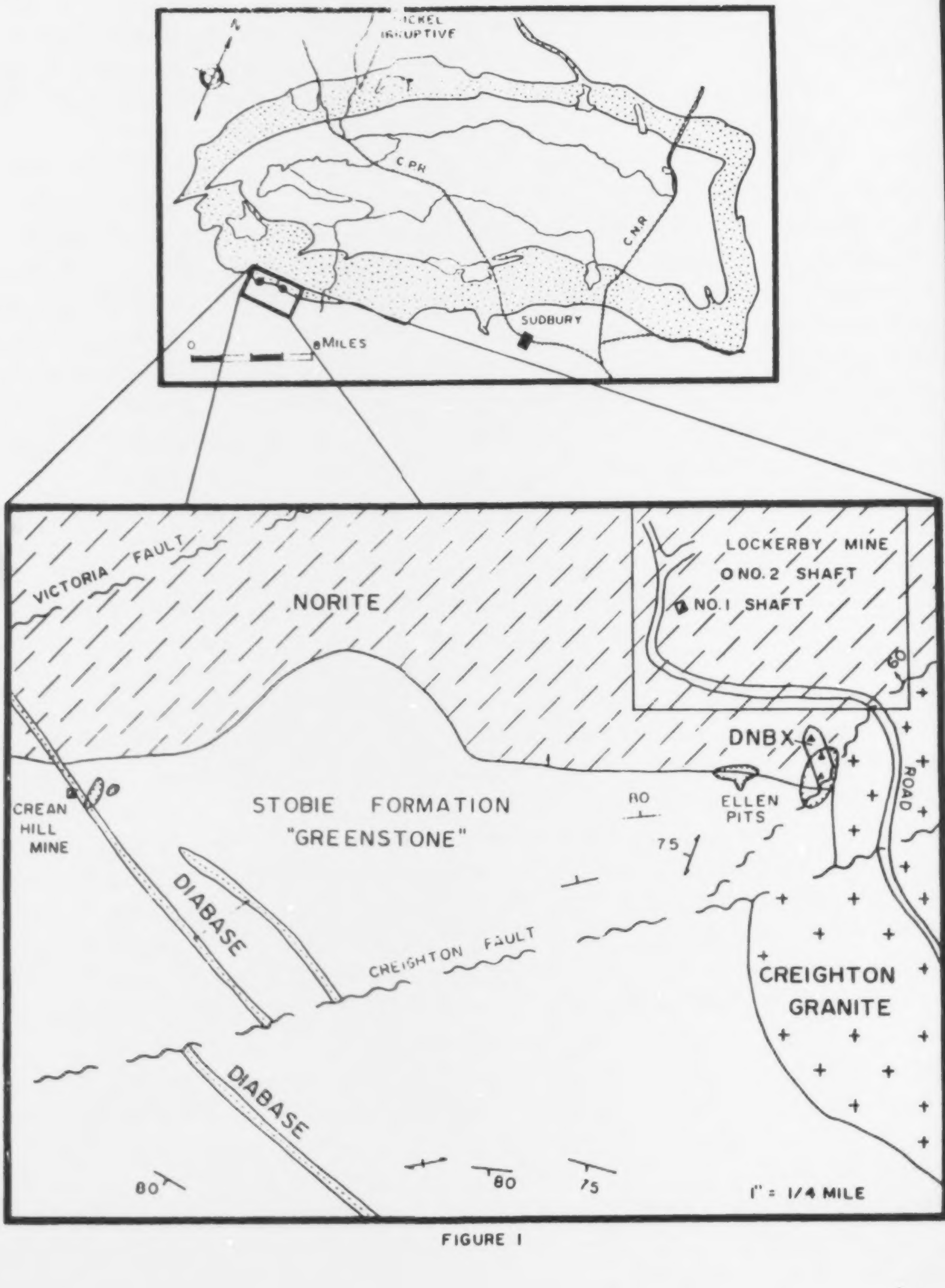

ICKEL
IRRUPTIVE
N
C.P.R.
C.N.R.
SUDBURY
0
8 MILES
VICTORIA FAULT
NORITE
LOCKERBY MINE
NO. 2 SHAFT
NO. 1 SHAFT
60
DNBX
ROAD
CREAN HILL MINE
STOBIE FORMATION "GREENSTONE"
80
ELLEN PITS
75
DIABASE
CREIGHTON FAULT
CREIGHTON GRANITE
DIABASE
80
80
75
1" = 1/4 MILE

FIGURE I

# LOCKERBY MINE

## LOCATION AND HISTORY

The Lockerby Mine is situated in the north half of Lot 2, Concession V of Denison Township on the southwest rim of the Sudbury Basin, twenty miles by road west of Sudbury. (Figure 1)

The property was originally held by the Lockerby family from 1888 to 1908, and then by the Dominion Bank until 1942, when it was purchased by Falconbridge Nickel Mines Limited. From 1916 to 1938, the property was optioned by a total of six mining companies who drilled ten holes near the south property boundary. During 1943, Falconbridge drilled three holes near the south boundary, and from 1961 to 1969, extensively drilled the remainder of the property. Shaft sinking commenced in 1969, underground drilling in 1970, and development production in 1974.

## GENERAL GEOLOGY

The orebody is located at a nose of the Sudbury Irruptive norite contact which protrudes into greenstone at the west end of the Creighton granite pluton. A major fault, which is part of, or subsidiary to the Creighton Fault, follows the east limb of the nose; this fault has been named the Main Fault. Within the nose, and abutting the fault is a breccia pipe. The sulphide ore occurs mainly as breccia and stringer sulphides within the fault zone, as blebs in the matrix of the breccia pipe, and disseminations in norite. The orebody consists of a single shoot which varies in width from a few feet to 150 feet, in length from 300 feet to 800 feet, and it has been defined over a depth of 4200 feet. The ore shoot outcrops at surface, where it has been mined by Inco's No. 1 Ellen pit, and dips northwesterly at 60° to Lockerby mine's 2250 level; having crossed the south property boundary at the 850 level. The shoot

... 2

dips almost vertically from the 2250 level to the 3475 level, and then the dip reverses to southeasterly at 70$^{0}$. The preceding features are illustrated in plan (Figures 2 and 3) and section (Figure 4).

## ROCK TYPES AND ASSOCIATED SULPHIDE MINERALIZATION

The compositions of the following rocks are not statistical averages, but estimates taken from diamond drill core and the occasional thin section. Accessory minerals are not quoted. Sulphides are essentially pyrrhotite-pentlandite-chalcopyrite.

### Norite

Generally, the norite is a medium to fine grained, dark grey to greenish-grey hypidiomorphic textured rock composed of 5-10% biotite, 55-65% labradorite, 25-30% amphibole, and 3-10% chlorite. Locally, 5-12% blue to glassy quartz occurs at the expense of other silicates. Pale grey rounded felsic fragments from ¼ inch to 2 inches in diameter are sometimes scattered throughout.

A 50 to 170 foot thick layer of medium to coarse grained light grey norite striking parallel to the east limb of the nose and dipping about 80$^{0}$ N., has been recognized east of the breccia pipe in the area where the main contact fault dips vertically. The composition of this norite appears similar to the general norite; however, the coarser feldspars give the rock a "felsic norite" appearance. Parts of this layer, expecially the footwall side, are heterogenous in grain size and contain clots of coarse grained noritic material which could be xenoliths. This layer is barren of sulphides, and relatively free of alteration zones.

Within the darker norite, there are spotty zones of strong chlorite

... 3

or amphibole alteration with variable amounts of disseminated sulphides. These zones occur both adjacent to the ore zone, the breccia pipe, and north of the "felsic norite" layer.

## Plagioclase-biotite Schist

A dynamically metamorphosed norite occurs as a discontinuous layer, usually less than 50 feet thick, between the norite proper and granite or greenstone. This fine grained, brownish-grey, gneissic to generally schistose rock is composed of 10% quartz, 30-60% labradorite, 15-40% biotite, 3-20% carbonate and 3-15% epidote-zoisite. The labradorite is partially altered to albite, and the biotite is after amphibole.

This schist was previously designated "feldspathic gneiss", and it appeared to be a rock intrusive into the norite along the Main Fault zone. However, the composition, and the many more recently observed gradational contacts with sheared norite, indicate a zone of altered norite which has been recrystallized locally to give a uniform fine grained dike-like appearance.

Granite fragments and lenses of quartz accompanied by coarse grained biotite schist are common in this rock. Sulphides occur as breccia sulphide lenses and stringers, and finely disseminated pyrrhotite-pentlandite, the latter mineral enhancing the ore grade in the thin eastern extremity of the ore shoot.

## Dark Norite Breccia

Rounded to subangular fragments of greenstone, ultramafics (amphibole schists),norite-like, and felsic rocks, occur in a fine grained dark norite matrix. The fragments vary in size from a fraction of an inch to several feet, but are generally 1 to 2 inches in diameter. Fragments

... 4

are usually closely packed, but may form as little as 10% of the rock, and in this extreme, the matrix is usually medium grained. The majority of the fragments are greenstone, about 10% are norite-like, and about 3% felsic. Norite-like fragments varying in size from 1 to 18 feet are scattered throughout, but are usually concentrated adjacent to the north contact of the pipe. Chlorite alteration is common. Sulphides occur throughout as blebs or rags interstitial to the fragments, or disseminated in norite-like fragments.

The dark norite matrix occasionally forms fragment free zones at the periphyry of the pipe, or forms dike-like protrusions outward from the pipe.

## Greenstone

Generally, the greenstone is a very fine grained, dark grey, finely banded rock with foliation parallel to the banding. It consists mainly of amphiboles with varying amounts of feldspars, quartz, biotite and chlorite. This rock has been classified as an amphibolite in a few thin sections. The fine bands alternate between mafic-rich with amphibole and felsic-rich with either plagioclase and amphibole or biotite-chlorite-quartz-sphene. The rock could be a metasediment, or a pyroclastic equivalent of basal Huronian volcanics.

Adjacent to the norite contact the banded amphibolite grades into a zone of massive porphyroblastic textured amphibolite with 2% quartz, 15% labradorite and 75% hornblende; the hornblende crystals ranging up to 15 mm in size. This zone varies from a few feet to 100 feet in width, and it probably represents a zone thermally metamorphased by the Sudbury Irruptive.

Some areas of the greenstone are an aphanitic, dark green, massive rock, usually highly fractured, and locally having a weak amygdaloidal or porphyritic fabric. These areas probably represent lava flows.

... 5

Sulphides, locally of ore grade, follow fault zones transecting the greenstone. These faults are generally part of the Main Fault zone.

## Creighton Granite

The granite is a medium to fine grained, pale grey to pinkish, gneissic rock, which is generally feldspar porphyritic. It is composed of 20% quartz, 50% K-feldspar, 20% albite, 5% biotite and 2% muscovite. Granitic appearing zenoliths, which are usually elongated parallel to the gneissosity, are scattered throughout, but appear to form only a small percentage of the rock. The gneissosity strikes northeasterly and dips between 50° to 80° southeasterly forming a trend tangential to the north contact of the granite.

Tongues of breccia sulphide often protrude into the granite, and chalcopyrite filled fractures are common within 100 feet of the main ore zone.

## Breccia Sulphide

The average breccia sulphide is composed of 40% rock fragments in a sulphide matrix, the fragments varying in size from a fraction of an inch to tens of feet. The fragments are generally granite or greenstone, depending upon which one of these rocks forms the south wall contact, and quartz-biotite schist, ultramafics, dark norite breccia and rarely norite. The sulphide matrix composition is about 80% pyrrhotite, 13% pentlandite, 8% chalcopyrite, minor pyrite, magnetite-ilmenite, and somtimes traces of sphalerite, galena, cubanite (?), and gersdorffite. None of the trace sulphides are recognizable megascopically. Breccia sulphide occurs as massive lenses up to 80 feet wide, veins, and stringers within the Main Fault zone.

... 6

## Structural Geology

The norite nose is most accentuated on the upper levels of the mine, and it becomes less prominent with depth. The Main Fault, a biotitic shear zone striking between N 45° E at surface to N.70° E below the 3650 level, follows, but does not necessarily form, the norite contact along the east limb of the nose. West of the nose, the fault divides with part striking southwesterly into the greenstone, and part following the greenstone-granite contact.

Another major shear zone, the Lockerby Fault, with an atitude of N 55°-70° E/45°-65° SE, joins the Main Fault just above the 3475 level, and probably contributes to the reversal in dip of the Main Fault below the 3475 level. Several other minor faults occur within the mine area, but they are generally difficult to trace because of abrupt changes in their intensity and offsetting by crosscutting fractures. One recognizable set of faults has an atitude of E-W/45°-70° S, and a 10 to 15 foot wide diabase dike has intruded one of these faults. This dike has been offset along the Main Fault by a left hand displacement.

The breccia pipe rakes due north from surface down to the 3475 level where it appears to have been truncated by the Lockerby Fault. However, narrow elongate portions of this breccia occur adjacent to the Main Fault down to the 3825 level. The pipe averages 300' x 350' in size, the longer axis striking N 35° W, and its shape varies amoeba-like from level to level. Dike-like protrusions emanate from the pipe, and these generally parallel the contacts of the norite nose. At the 3475 level, a smaller pipe branches from the main pipe and parallels it 200 feet to the west up to the surface (see Figure 3). The two pipes are apparently joined by a horizontal breccia pipe between the 1700 and 1875 levels. The origin of the breccia pipe is uncertain, but it appears to be intrusive into the norite.

... 7

A very strong conjugate joint system is prominent within the norite. The sets have atitudes of N 65° W/65°-85° SW and N-S/70°-90° W. These joints are generally filled with quartz-carbonate-chlorite (or biotite), and in conjunction with zones of N 75° E/20°-30° NW low angle joints, they create ground problems in mining.

Several diabase dikes from a few inches to a few feet in width transect all the rocks. Several of these dikes strike N 55° E and dip either moderately southeast or steeply northwest. Diabase dikes in granite are usually altered to biotite schist.

Narrow granite dikes without chilled selvedges intrude the rocks lying to the north of the granite pluton. These dikes may represent a remobilization of the Creighton Granite by the intrusion of the Sudbury Irruptive.

## Ore Zones

Ore grade sulphides are divided into the Main Ore Zone and North Ore Zones. The Main Zone comprises breccia sulphide and adjacent sulphide stringers and disseminations in all rock types transected by the Main Fault. The intensity of sulphide mineralization decreases northerly from the fault within dark norite breccia and to a lesser extent in norite. Irregular shoots of low grade ore occur within the dark norite breccia and embayments of norite into the breccia pipe to the north of the Main Fault, and these shoots have been designated as North Zones; they comprise about 5% of the orebody.

The Main Fault and breccia pipe appear to be the structures which have localized the ore forming fluid at Lockerby Mine.

## MINE PLANS AND SECTION

## LEGEND

| | |
|---|---|
| ▥ | ORE |
| + | NORITE |
| ■ | SCHIST |
| ⁻_ | PLAGIOCLASE-BIOTITE SCHIST |
| △ | DARK NORITE BRECCIA |
| ▧ | GREENSTONE |
| π | GRANITE |
| ≡ | DIABASE |
| ░ | BRECCIA SULPHIDES |
| ⊤ ⌊ | QUARTZ |
| ∽ | FAULT |

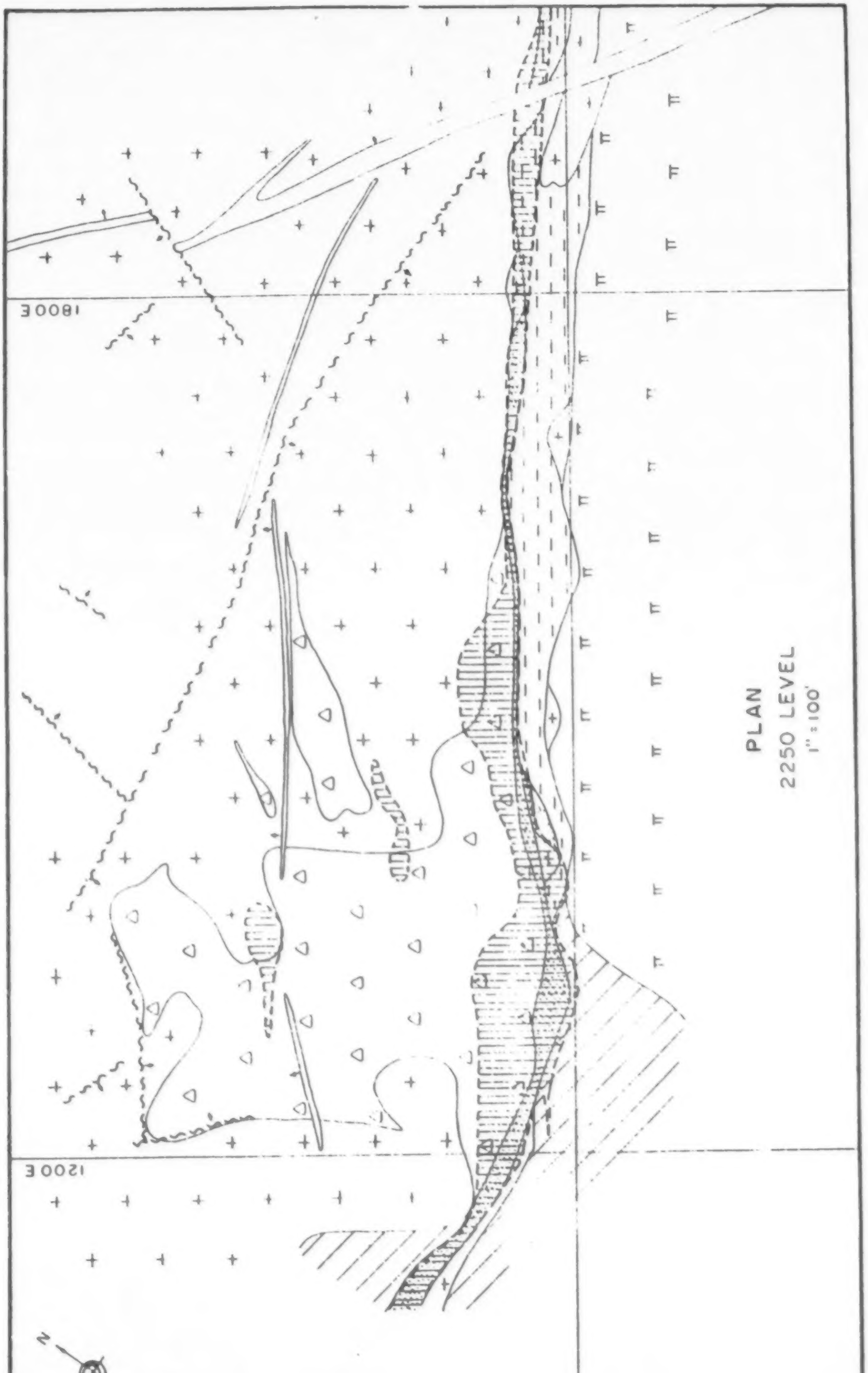

PLAN
2250 LEVEL
1":100'
1800E
1200E
N
CL

FIGURE 2

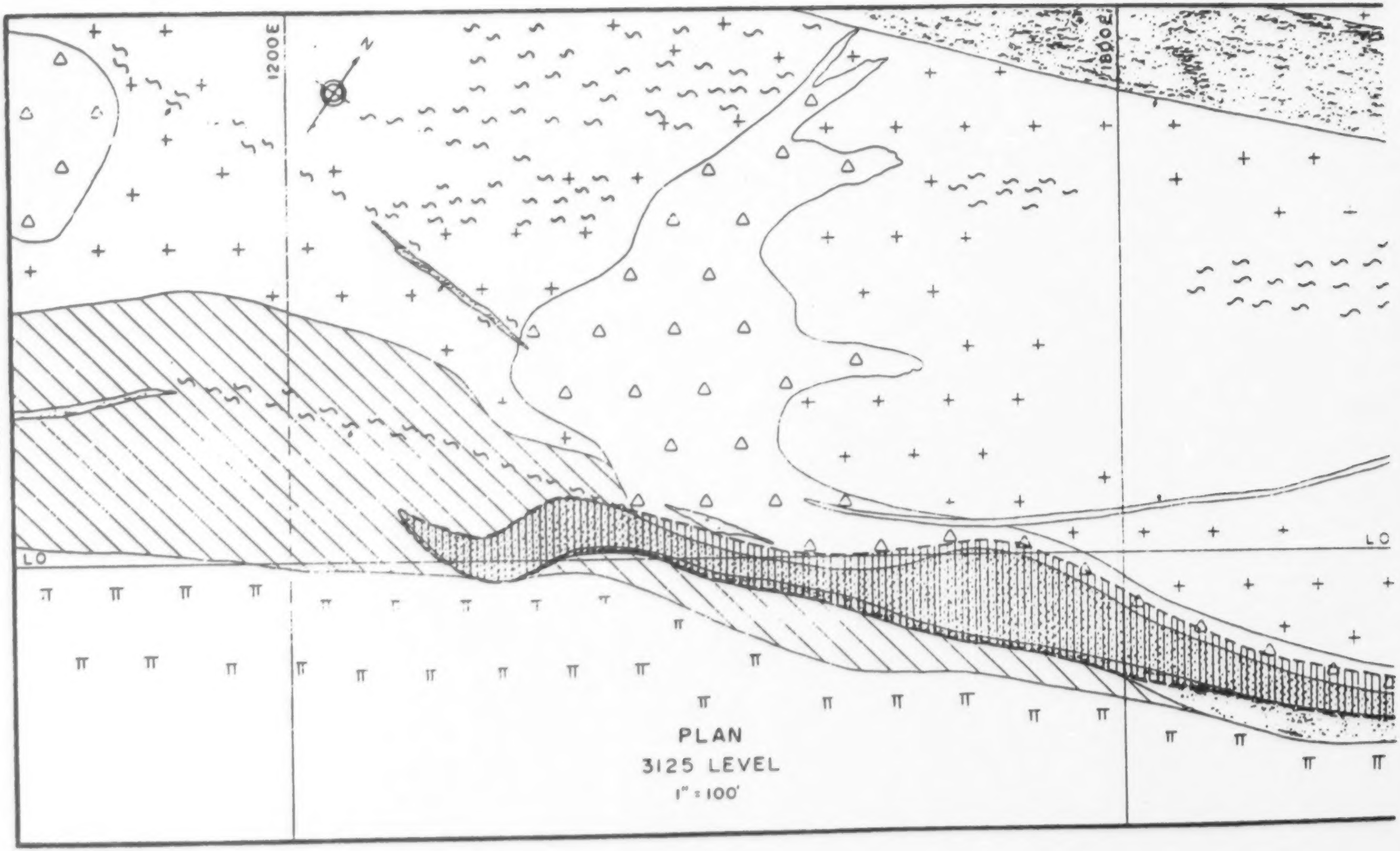
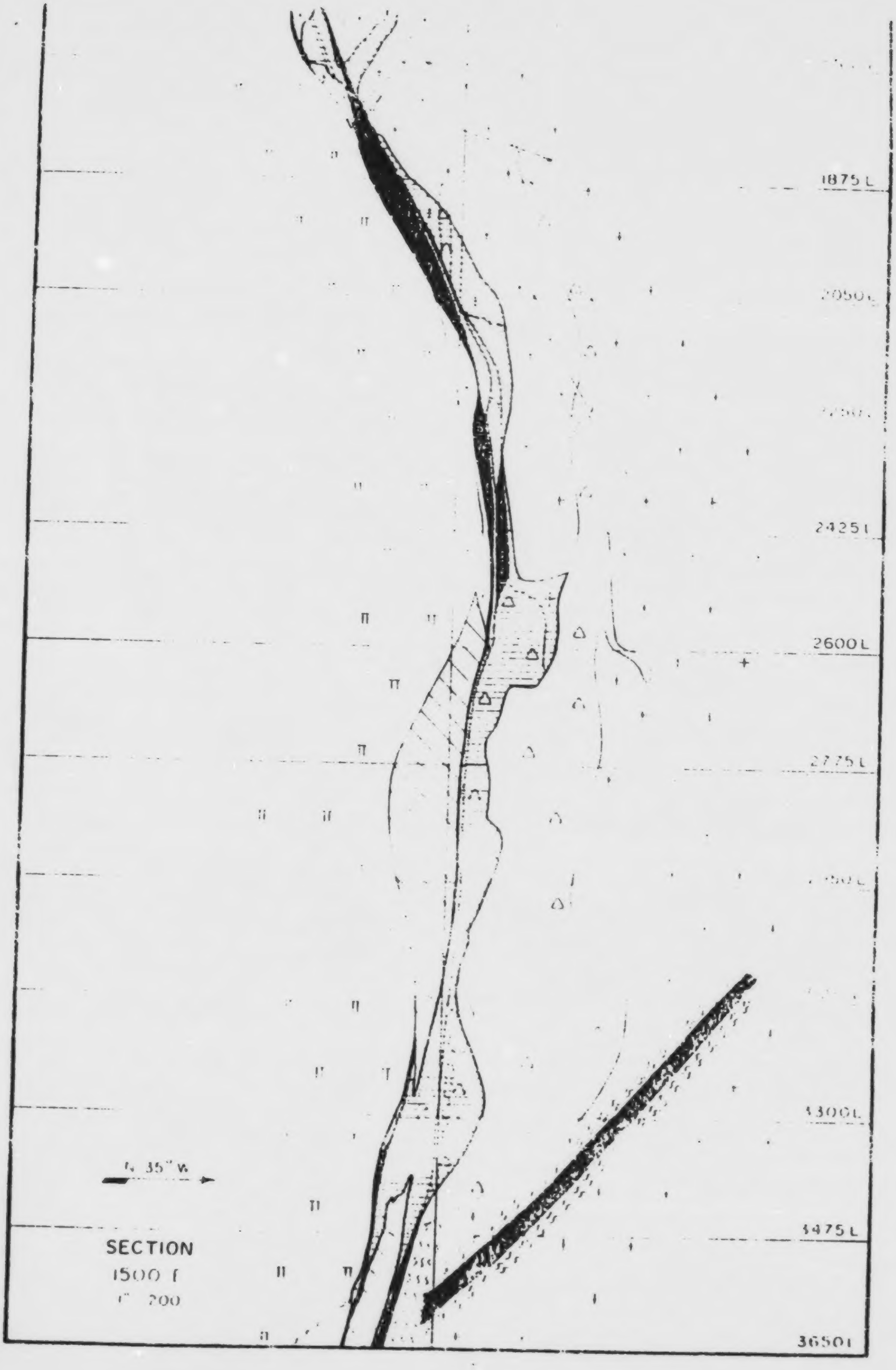

PLAN
3125 LEVEL
1" = 100'
1200E
1800E
LO
LO
N

FIGURE 3


1875 L
2050 L
2250 L
2425 L
2600 L
2775 L
2950 L
3300 L
3475 L
3650 L
N 35° W
SECTION
1500 E
1" = 200'

## SUMMARY OF STOPING METHODS

### 1. TRANSVERSE BLAST HOLE

Dimensions: 36' wide, 50' to 150' deep, 150' to 200' in height.

Access: trackless cross-cuts from sub levels to stope width overcut and undercut.

Slot Raise: 7' x 7' Alimak raise at north end of stope.

Blastholes: vertical to -80° holes drilled with an "In-the-Hole" hammer using 6½" button bit. Three transverse lines of holes spaced at 15'. Centre line holes have an 11' burden; outer line holes a 5½' burden. Efficiency: 45'/manshift. Hole deviation 2%.

Blasting: Five holes loaded with Anfo for each blast (about 5000 to 6000 tons). Blasts fired from surface. Efficiency: 3 manshifts to load and fire a blast.

Ore Removal: Mucking from undercut to orepass with LHD equipment. Final muckpile removed by remote control Cavo mucking machine.

Fill: cemented tailing fill.

### 2. LONGITUDINAL TIGHT CUT AND FILL RILL

Dimensions: 14' to 50' wide, 150 to 250' in length.

Access: trackless cross-cuts driven to stope width sub levels at 50' to 62' vertical intervals. Access to cross-cuts from ramp or service raise.

Slot Raise: 7' x 7' boxhole raise at end away from access.

Blastholes: 60° up holes drilled by a "long-hole" drill using 2" or 2 1/8" cross bits. Holes spaced at 4' centres with a 3½' burden.

Blasting: Two rows of holes loaded with Anfo for each blast.

Ore Removal: Mucking from undercut sub level to ore pass with LHD equipment.

Fill: Waste rock dumped into void from overcut sub level after removal of ore from each blast.

### 3. LONGITUDINAL CUT AND FILL

Dimensions: 11' minimum mining width with LHD equipment. Post pillars (20' x 20') required over 50' in width. 32' vertical sill pillar with 144' vertical stope above (level interval of 176')

Access: Service raises and mill holes from track driven cross-cuts.

Mining: 10' vertical cuts taken by jackleg breasting or wagon drill "uppers."

3. LONGITUDINAL CUT AND FILL (Continued)

Ore Removal: Cavo LHD equipment in stope moves broken ore to millholes. Trains on levels move ore from chutes to orepass.

Fill: Cemented tailing fill.

4. RAMP ACCESS CUT AND FILL

Similar to the preceding except that the access is from ramp sub levels at 50' vertical intervals. Cross-cuts are driven down to a cut 25' below the sub level elevation, and then the cross-cuts are raised through a shrinkage system for each 10' cut until they reach an elevation 25' above the sub level elevation. Ore removal is by LHD equipment through the cross-cuts to the ore pass.

JJP/fb

## SUMMARY OF MINE STATISTICS

### 1. SHAFTS

(a) No. 1 (Production) Shaft - 14'5¼" x 14'5¼" four compartment with timber sets. Depth 4422'. Conventional 14' drum hoist operating two counterbalanced cage over skip conveyances at a speed of 2200 FPM. Skip capacity 9 tons of ore.

(b) No. 2 (Service) Shaft - 20'6" diameter concrete lined four compartment with steel sets. Depth 4165'. Friction hoist operating one cage at a speed of 2000 FPM. Cage capacity 44 men or 21,000 lbs. Services 21 levels.

(c) Ventilation Shaft - 15'0" diameter, concrete lined, single open compartment. Depth 3625'.

### 2. STOPING METHODS

(a) Transverse Blast Hole.

(b) Longitudinal Tight Cut and Fill Rill.

(c) Longitudinal Cut and Fill (Track Serviced).

(d) Ramp Access Cut and Fill.

### 3. ORE TRANSPORTATION

(a) Underground: stope to ore pass by 5 or 8 cubic yard trackless LHD equipment or by 93 cubic foot Granby cars hauled by battery locomotives. Single 7' diameter ore pass feeds a 48" x 60" jaw crusher on the 4000 level. Crusher discharges into a 2700 ton bin which feeds a conveyor belt system to the No. 1 shaft loading pocket just below the 4325 level.

(b) Surface: hoisted ore is dumped into a 200 ton bin in No. 1 shaft headframe. This bin feeds a conveyor belt which transports the ore to a 3500 ton truck loadout storage bin. The ore is trucked 34 miles to the Strathcona Mill.

### 4. FILL

Tailing fill is transported by truck from Strathcona Mill to the Tailing Fill Plant at Lockerby. The dry fill is mixed with water and cement, and pumped hydraulically to the underground levels through a series of 3½" and 4½" drill holes. Level pipe lines transport the fill to the stopes. A 20:1 ratio of tailing sand to cement is generally used for stope fill, with other ratios for special purposes.

### 5. VENTILATION

Fresh air is downcast through No. 2 shaft from the 125 level intake raise, and upcast through the ventilation shaft to the 125 level exhaust raise. An 8'3" diameter, 400 HP fan on the 125 level, with a capacity of 360,000 cfm, operates the system with the aid of smaller underground booster fans. During winter operations the air is heated by three

... 2

## 5. VENTILATION (Continued)

8,000,000 BTU propane gas heaters which have their air supplied by 6' diameter, 60 HP, 120,000 cfm capacity fans.

## 6. WATER SUPPLY

Water is obtained from the Vermillion River via an8000 foot, 8" polyethylene pipeline.

## 7. COMPRESSED AIR SUPPLY

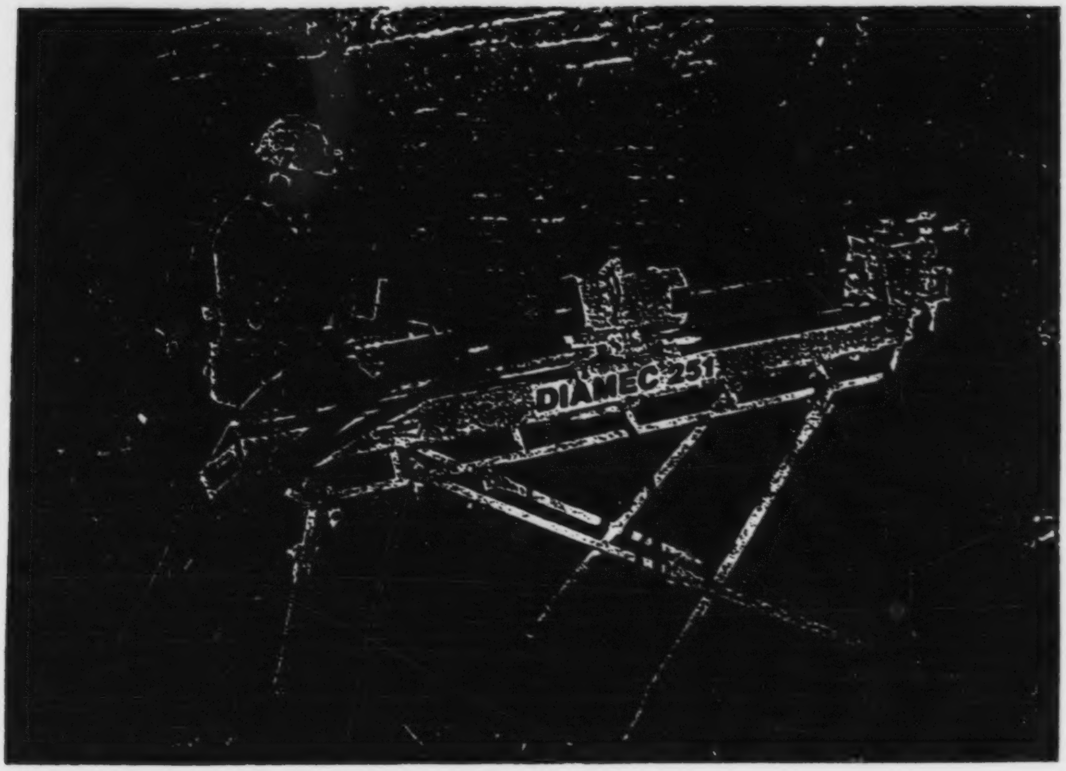
A 4190 cfm C.I.R. reciprocating compressor and a 6970 cfm C.I.R. Centac type compressor supply air to the mine and plant at a pressure of 110 psi.

JJP/fb

# APPENDIX B

## CRAELIUS DRILL DATA

# Craelius Diamond Core Drill DIAMEC® 251



*DIAMEC 251 - Electric variant for underground applications*

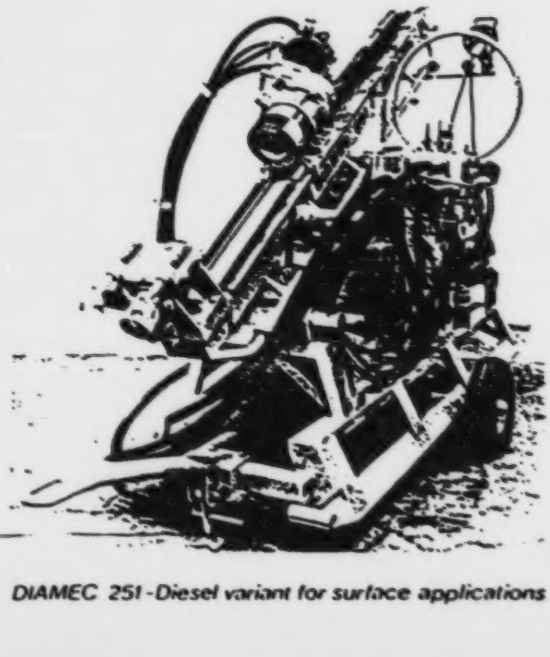
DIAMEC 251 is an all-hydraulic core drill designed on the basis of practical experiences obtained with our well-known DIAMEC 250.

DIAMEC 251 has the capacity to drill holes down to a depth of 400 m with Ø 43 mm aluminium rods.

For torque-demanding drilling with low rpm, (alt. spindle speed) a high torque gearbox is available as optional equipment (see page 8). Following adaptation, this gives the possibility of drilling with rod dimensions up to Ø 114 mm.

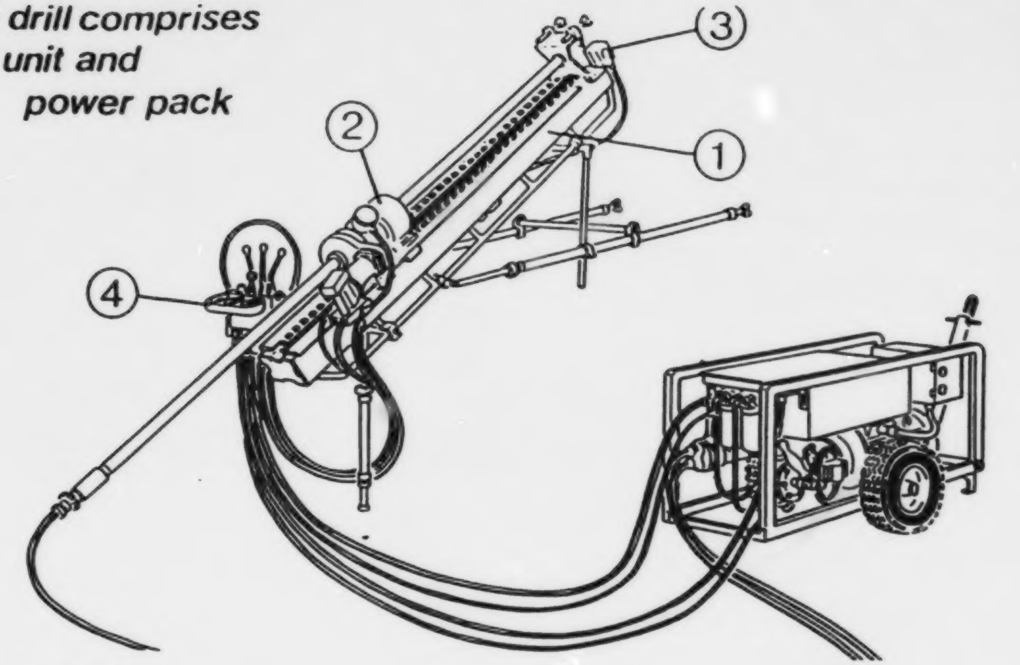
DIAMEC 251

- can be supplied either with an electric power pack for drilling underground, or as a diesel driven variant for surface drilling.
- is simple to transport, primarily due to the component design of the entire unit and through the mobile power pack supported on tyred wheels.
- requires a minimum of operating space, through the overall compact design of the unit and its small dimensions.
- is easy to handle and operate, thanks to the ergonomic and well-proven design of the control panel.
- has a hydraulic system equally suited to HF-B and HF-C fluids as well as mineral oils.



*DIAMEC 251 - Diesel variant for surface applications*

# *The core drill comprises a drilling unit and power pack*



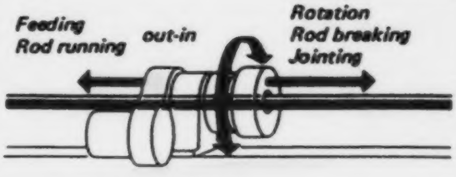
**The drilling unit consists of a:**

- **① Feed frame**
- **② Rotation unit**
- **③ Rod holder**
- **④ Control panel**

1) The feed frame is available in two lengths, 850 and 1600 mm respectively. The frame has support legs for achieving stable set-ups in different drilling directions. Movement of the feed is obtained through a double-acting hydraulic cylinder, where movement of the piston is transferred via a chain, to the cradle of the rotation unit. Max feed force is 41.3 kN (4130 kp).

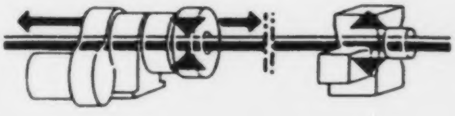
The feed frame anchors to the rock face with an expansion bolt. The hole for the bolt can be drilled with the core drill.

2) The rotation unit comprises a hydraulic motor, gear box and chuck. When the hydraulic chuck is activated, oil pressure is transferred to the jaws in the chuck which press against the drill rod. When oil pressure ceases, the chuck opens automatically through spring pressure. The hydraulic motor is pressure-compensated and develops constant power output at a spindle speed range between 675–2200 rpm, at a working pressure of 250 bar. The spindle speed is steplessly variable. Through simple replacement of the gear, the gear ratio can be altered from 1.5:1 (standard) to 2.5:1, which then produces a spindle speed of between 410–1350 rpm.

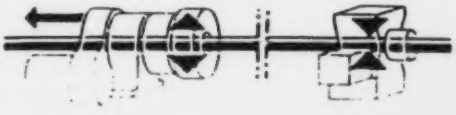


*During rod running, alternate movements of the rotation unit and rod holder occur. When one opens, the other automatically retains.*

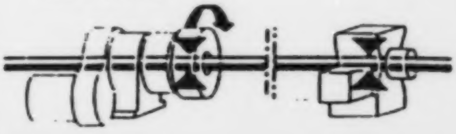
*Rod running out — in*

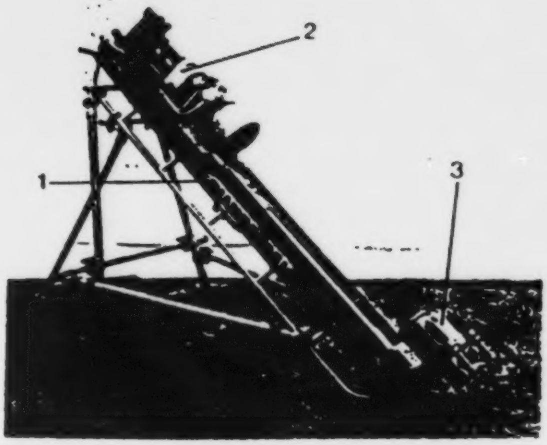


*Return of the rotation unit*



*Rod breaking*





3) The rod holder is a hydraulic/mechanical chuck, whereby the method of operation is the reverse to that of the hydraulic chuck.

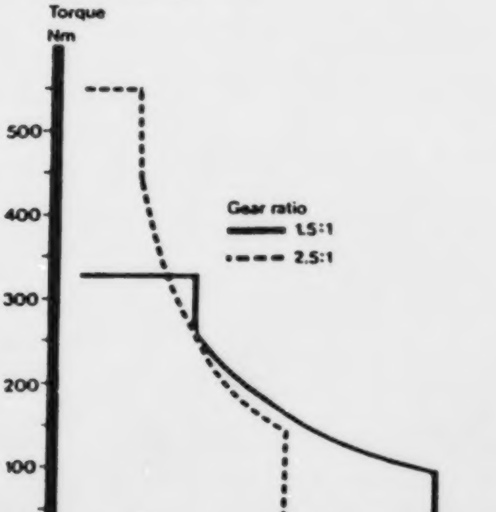
The rod holder grips mechanically with the aid of spring pressure that directly influences the jaws. If, for some reason or other, hydraulic pressure should suddenly cease, then the rod holder grips the drill string and retains it in a firm grip.

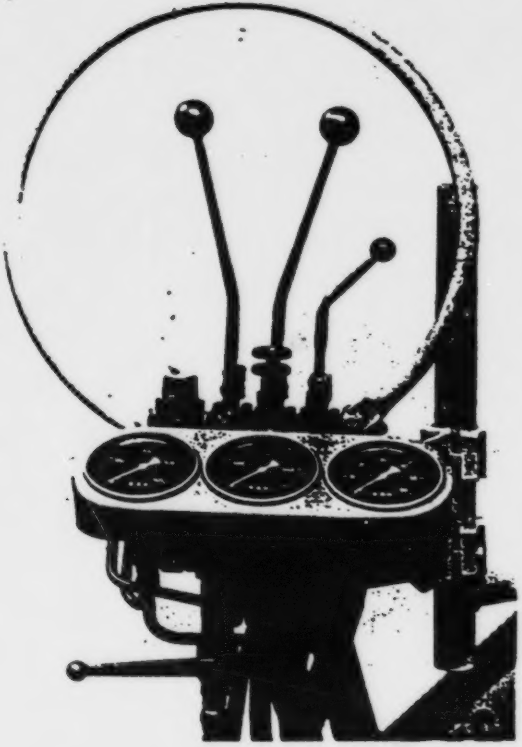
Through the drill string being held throughout the entire work cycle, either by the rotation unit or by the rod holder, the risk of losing a drill string down a hole is minimal. This system also serves as an additional safety feature for the driller, e.g. when drilling acute-angled upward holes.

4) The control panel is ergonomically designed to function with optimal effect, safety and ease in operation. The arrangement of the hydraulic hoses is so designed that the panel can be positioned at a safe distance from rotating components. Rotation, feeding and rod running are controlled with three levers, feed pressure is regulated with a handwheel. These four controls guide and operate all drilling and rod running functions. The minimal number of controls makes drilling a simple and rapid process to learn.

Torque-rotation speed diagram
(Efficiency rate 1.0)

With standard rotation unit





The control panel is also equipped with pressure gauges that indicate system pressure, feed force and flushing water pressure.

The panel is normally mounted at the side of the feed frame, however, for practical purposes it may be more suitable or convenient to place it elsewhere or at a distance.

## *Simple drilling sequences*

The driller controls all drilling functions from the control panel, using just three control levers and a control valve. Manual rod handling merely entails lifting away alternatively inserting a new drill rod onto the feed frame.

Rod handling occurs mechanically with the aid of the rotation unit and the rod holder. The rotation unit performs movements of the feed and rotation. The rod holder retains the drill string while the rotation unit returns to the start position to continue new feed movement in or out of the drill hole. During rod running, the chuck functions of the rotation unit and rod holder are automatically synchronised, so that the rod holder grips when the rotation unit opens, and vice versa.

The rod holder also locks the drill string when the rotation unit is carrying out rod breaking and jointing.

The hydraulic system is designed for various types of hydraulic fluids and mineral oils as well as HF-B and HF-C fluids.

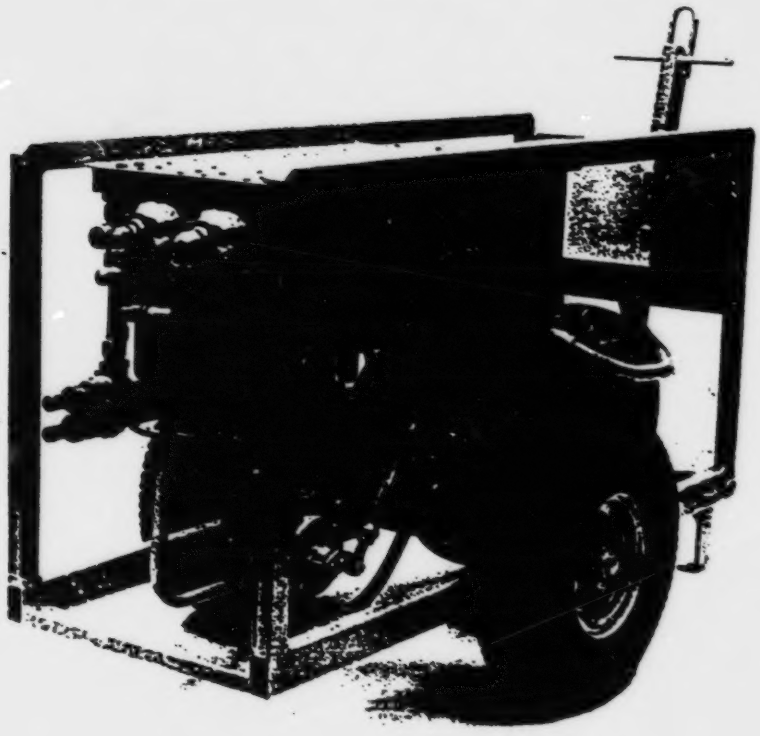
The double hydraulic pump of the power pack produces a system pressure of 250 bar at 65 l/min. One of the system's pumps supplies pressure and flow to the feed circuit while the other serves a similar function to the rotation circuit. Through this circuit distribution it is possible to maintain a constant feed force, independent of the speed of rotation and torque.

### *POWER PACK 30E*

This comprises an electric motor (30 kW), a pressure-compensated, double hydraulic pump and a hydraulic oil tank.

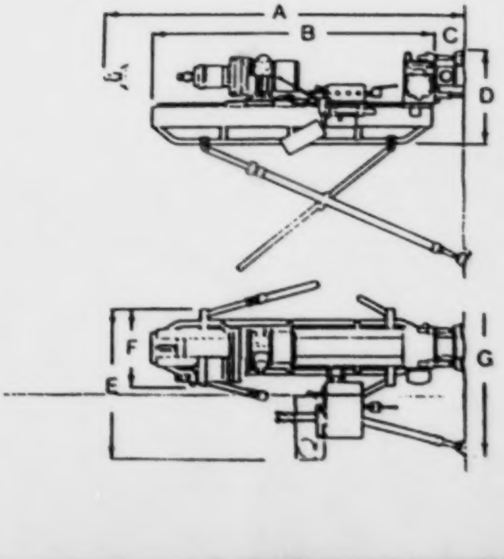
The hydraulic oil is cooled with a water-cooled oil cooler. Separate drainage and return oil filters ensure the hydraulic oil is purified and cleansed before it re-enters the oil tank. The filter housing has indicators that indicate when it is necessary to change a filter cartridge.

The power pack can be positioned separate and independent of the other equipment, only being limited by the length of the hydraulic hoses (4.5 m standard, 25 m optional equipment).



## Data

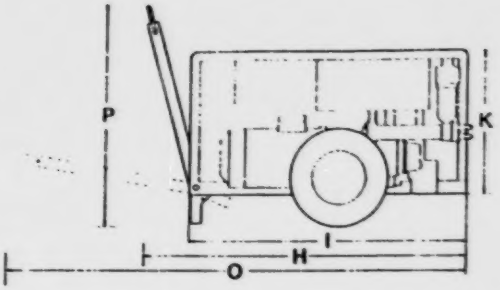
**Dimensions, mm (in.)**

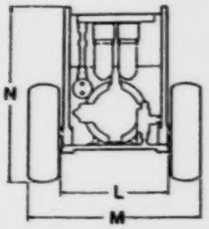


Drill unit

| | Drill unit with feed frame type 850 | Drill unit with feed frame type 1600 |
|---|---|---|
| A | 1940 (76) | 2690 (106) |
| B | 1540 (61) | 2290 (90) |
| C | 170 (7) | 170 (7) |
| D | 520 (20.5) | 520 (20.5) |
| E | 820 (32.5) | 820 (32.5) |
| F | 420 (16.5) | 420 (16.5) |
| G | 1160 (46) | 1160 (46) |

## Power pack





| | Power pack 30E | | |
|---|---|---|---|
| H | 1850 (73) | M | 975 (38) |
| I | 1600 (63) | N | 1000 (39) |
| K | 820 (33) | O | 2700 (106) |
| L | 595 (23) | P | 1260 (50) |

## Weights, kg (lb)

| | |
|---|---|
| Rotation unit | 95 (209) |
| Rod holder | 55 (121) |
| Feed frame 850 | 72 (159) |
| Feed frame 1600 | 107 (236) |
| Mounting frame | 41 (90) |
| Control panel with hydraulic hoses | 65 (143) |
| Power pack 30E, without oil | 520 (1145) |

## Capacity

| Drilling depth | Hole diameters | Drill rods |
|---|---|---|
| 550 m (1810 ft) | 36 mm, E | 33 mm, EW alu rods |
| 270 m (890 ft) | 36 mm, E | 33 mm, EW steel rods |
| 400 m (1315 ft) | 46 mm, A | 43 mm, AW alu rods |
| 210 m (690 ft) | 46 mm, A | 42 mm, AW steel rods |
| 270 m (890 ft) | 56 mm, B | 53 mm, BW alu rods |
| 160 m (530 ft) | 56 mm, B | 50 mm, BW steel rods |

## Rod holder

| | |
|---|---|
| Inner diameter | 58 mm |
| Axial holding force | 12 kN |
| Guide sleeve diameter | 43 mm |
| Set of 2 jaws for diameter | 43 mm |
| Expansion bolt, size | 19x1000 mm |

### Optional equipment

| | |
|---|---|
| Guide sleeve diameter | 33 mm |
| Set of 2 jaws for diameter | 33 or 53 mm |

## Rotation unit

| | |
|---|---|
| Inner diameter | 58 mm |
| Spindle speed | 200–2200 rpm |
| Gear ratio | 1.5:1 or 2.5:1 |
| Hydraulic motor | |
| Working pressure | 200 bar**) |
| Max pressure | 250 bar**) |
| Oil flow, max. | 1 l/s |
| Chuck | |
| Axial holding force | 30 kN at 250 bar**) |
| Guide sleeve diameter | 43 mm |
| Set of 4 jaws for diameter | 43 mm |

### Optional equipment

| | |
|---|---|
| Guide sleeve diameter | 33 mm |
| Set of 4 jaws for diameter | 33 or 53 mm |

## Feed frame

| | |
|---|---|
| Feed length | 850 or 1600 mm |
| Feed force*) | 41.3 kN |
| Pull, max.*) | 31.5 kN |
| Feed rate, max. | 0.33 m/s |

## Power pack 30E

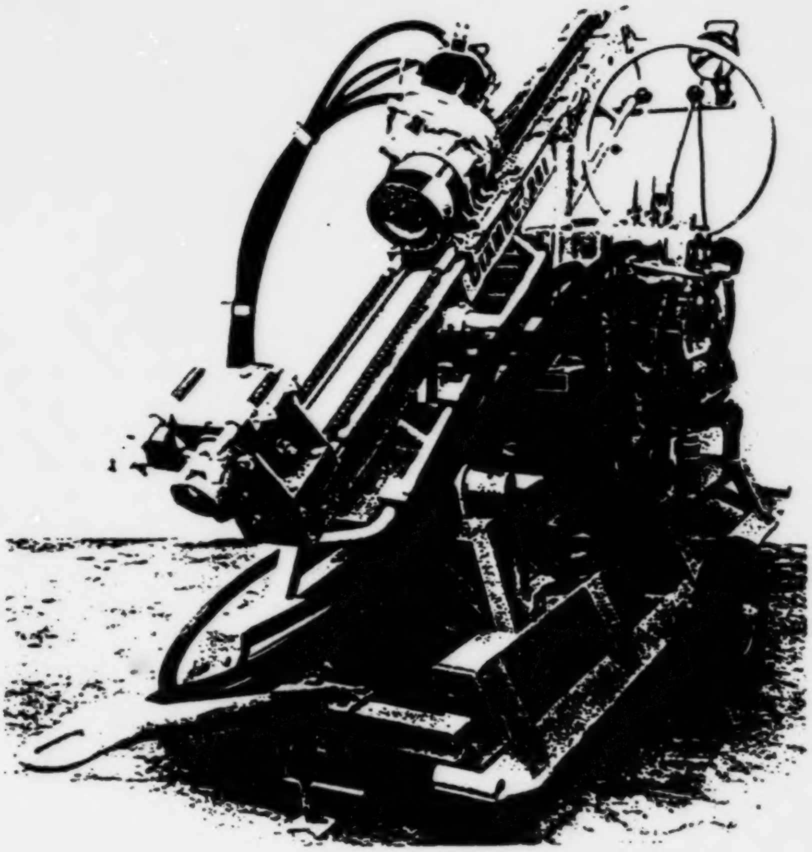
| | |
|---|---|
| Electric motor ASEA M200L (IM 2001), 4 pole | 30 kW |
| Hydraulic main pump Bosch Radial 63 $cm^3$/True disp. | 40 $cm^3$ |
| Working pressure | 200 bar**) |
| Max. pressure | 250 bar**) |
| Oil flow, max. | 1 l/s |
| Hydraulic service pump Bosch Radial | 16 $cm^3$ |
| Working pressure | 200 bar**) |
| Max. pressure | 250 bar**) |
| Oil flow, max. | 0.33 l/s |
| Oil, operating temperature | 50°–80°C |
| Hydraulic tank, volume | 70 l |
| Filters, filtration rate, absolute | 3 μ |
| Hydraulic hoses, length | 4.5 m |
| Wheel dimensions | 550x168 mm |

---

*) Theoretical value. Average degree of efficiency 0.70 – 0.75

**) 1 bar = 100 kPa = 1.02 $kg/cm^2$

# DIAMEC 251, diesel version, a compact drilling unit for surface applications



Available as a:

**Standard variant** – compact unit mounted on a single, rubber-tyred skid frame.

**Separate unit** – drilling unit mounted on one skid frame and the power pack mounted on a second skid frame.

The same drilling unit is used for both the diesel and electrical variants. The frame for the diesel variant however, has no support legs.

*Power pack dD4*

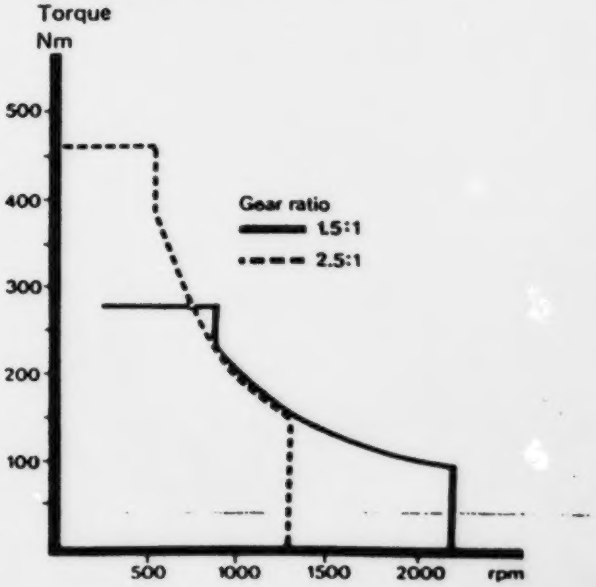
This comprises a 4 cylinder, air-cooled diesel motor developing 40 kW at 1800 rpm.

The hydraulic pump supplies a system pressure of max. 210 bar at 1.2 l/s. Max. working pressure is 175 bar. The hydraulic motor of the rotation unit operates within a wider angular range which gives a displacement/rpm ratio with higher torque and broader rpm register.

**Torque-Rotation speed**

**(Efficiency rate 1.0)**

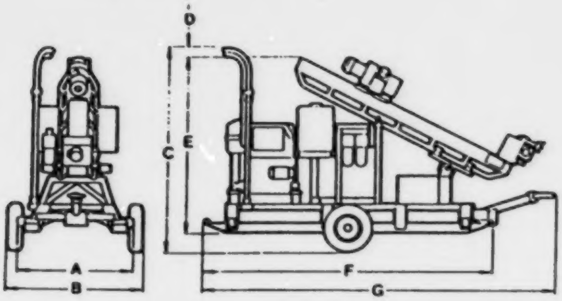
**with standard rotation unit**



# Data

## Dimensions

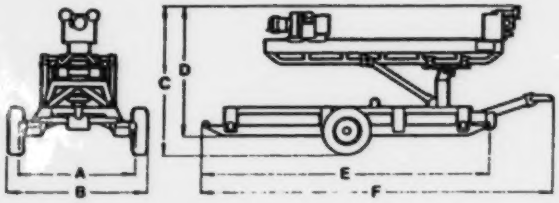
### Standard variant



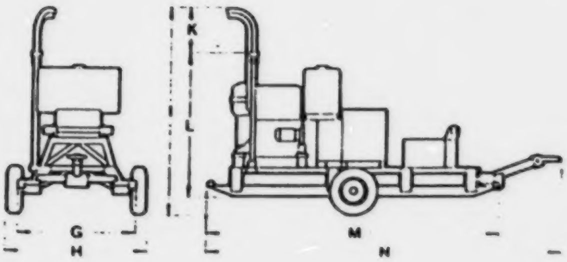
| | mm | in. | | mm | in. |
|---|---|---|---|---|---|
| A | 1090 | 43 | E | 1650 | 65 |
| B | 1340 | 53 | F | 2700 | 106 |
| C | 1940 | 76 | G | 3480 | 137 |
| D | 100 | 4 | | | |

### Separate unit

**Drill unit**



**Power pack**



| | mm | in. | | mm | in. |
|---|---|---|---|---|---|
| A | 1090 | 43 | H | 1340 | 53 |
| B | 1340 | 53 | I | 1940 | 76 |
| C | 1390 | 55 | K | 400 | 16 |
| D | 1200 | 47.5 | L | 1350 | 53 |
| E | 2700 | 106 | M | 2700 | 106 |
| F | 3480 | 137 | N | 3480 | 137 |
| G | 1090 | 43 | | | |

**Weights, kg (lb)**

| | |
|---|---|
| Standard variant with one skid frame incl. wheels | 1530 (3370) |
| Separate unit with two skid frames incl. wheels | 1860 (4100) |
| Drill unit | 650 (1432) |
| Power pack | 1210 (2665) |

## Capacity

| Drilling depth | Hole diameters | Drill rods |
|---|---|---|
| 550 m (1810 ft) | 36 mm, E | 33 mm, EW alu rods |
| 270 m (890 ft) | 36 mm, E | 33 mm, EW steel rods |
| 400 m (1315 ft) | 46 mm, A | 43 mm, AW alu rods |
| 210 m (690 ft) | 46 mm, A | 42 mm, AW steel rods |
| 270 m (890 ft) | 56 mm, B | 53 mm, BW alu rods |
| 160 m (530 ft) | 56 mm, B | 50 mm, BW steel rods |

## Rod holder

| | |
|---|---|
| Inner diameter | 58 mm |
| Axial holding force | 12 kN |
| Guide sleeve diameter | 43 mm |
| Set of two jaws for diameter | 43 mm |
| Expansion bolts, size | 19x1000 mm |
| **Optional equipment** | |
| Guide sleeve diameter | 33 mm |
| Set of two jaws for diameter | 33 or 53 mm |

## Rotation unit

| | |
|---|---|
| Inner diameter | 58 mm |
| Spindle speed | 200–2200 rpm |
| Gear ratio | 1.5:1 or 2.5:1 |
| Hydraulic motor | |
| Working pressure | 175 bar**) |
| Max. pressure | 210 bar**) |
| Oil flow, max. | 1.2 l/s |
| Chuck | |
| Axial holding force | 50 kN at 210 bar**) |
| Guide sleeve diameter | 43 mm |
| Set of 4 jaws for diameter | 43 mm |
| Optional equipment | |
| Guide sleeve diameter | 33 mm |
| Set of 4 jaws for diameter | 33 or 53 mm |

## Feed frame

| | |
|---|---|
| Feed length | 1600 mm |
| Feed force*) | 34.7 kN |
| Pull, max.*) | 26.4 kN |
| Feed rate, max. | 0.33 m/s |

## Power pack dD4

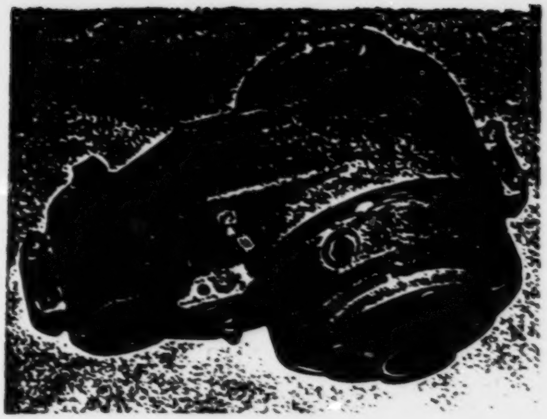
| | |
|---|---|
| 4-cylinder diesel engine Deutz F4L 912 rating at 1800 rpm | 40 kW |
| Hydraulic pump | |
| Working pressure | 175 bar**) |
| Max. pressure | 210 bar**) |
| Oil flow, max. | 1.2 l/s |
| Hydraulic tank, volume | 65 l |
| Filters, filtration rate, absolute | 3 µ |
| Hydraulic hoses, length | 4.5 m |
| Wheel size | 460x138 mm |

---

*) Theoretical value. Average degree of efficiency 0.70 – 0.75

**) 1 bar = 100 kPa = 1.02 kg/cm²

# High torque gearbox



For applications that require high torque and a lower rpm, Diamec 251 can be equipped with a high torque gearbox, designed to be mounted on the cradle of the rotation unit.

Equipped with the high torque gearbox, drilling can be carried out with rod dimensions up to Ø 114 mm. The gear ratio 4:1 produces a maximum torque of 180 kpm at 170 rpm. Use of the higher torque and lower rpm is particularly advantageous when drilling down casings and with down-the-hole equipment.

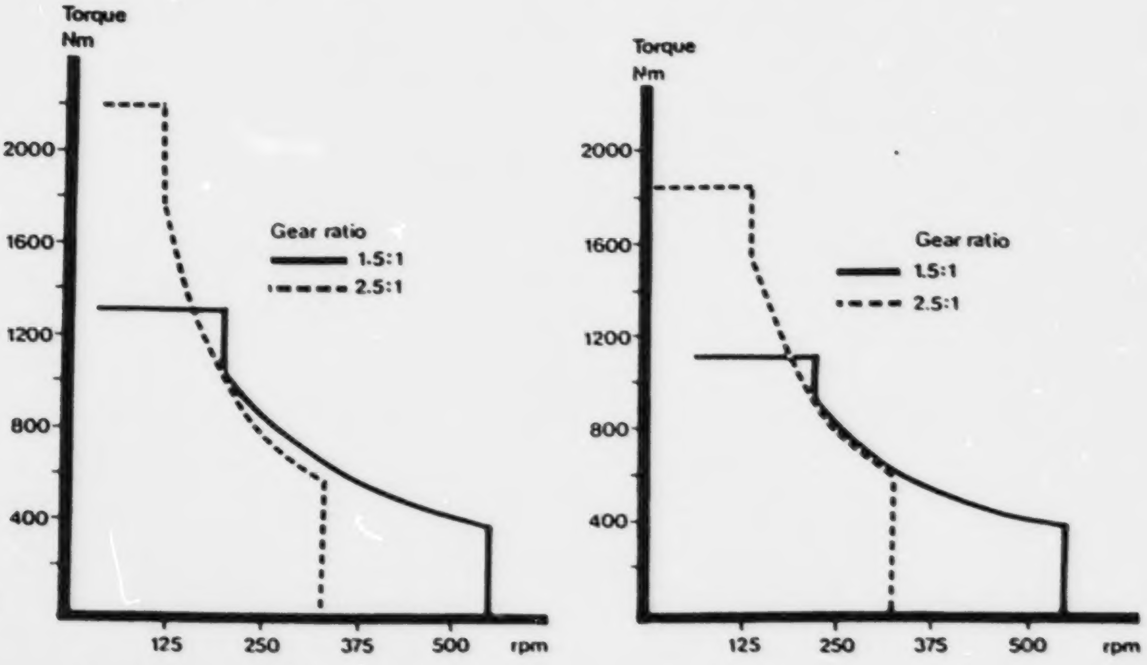
## DATA

Complete Equipment for DIAMEC 251 Comprises:
High torque gearbox, compl.
– Mounting assembly, compl.
Conversion assembly for rotation unit
Conversion set for rod holder

**High torque gearbox**

| | |
|---|---|
| Ratio | 4:1 |
| Transmission | Triplex chain |
| Lubrication | Grease |
| Chuck | 4-jaw mechanical |
| – max. inner diam. | 114 mm |
| Weight | 65 kg |

Torque/rpm for high torque gearbox



ELECTRIC-POWERED VARIANTS

DIESEL DRIVEN VARIANTS

1 Nm = 0.102 kgm = 0.738 ft.lb

Craelius

*Craelius products are sold and serviced worldwide through own subsidiar agents*

Craelius AB, Box 504, S-195 01 Marsta, Sweden
Telephone 0760-180 00. Telex 152 90 diabor s

**CRAELIUS DIVISION**
Unicom Abrasives of Canada Limited
192 Pearl Street, East,
Brockville, Ontario

# APPENDIX C

## TROW ONTARIO LIMITED REPORT

Trow

# RESEARCH DRILLING & TESTING PROGRAM
# CLASSIFIED MILL TAILINGS BACKFILL
# LOCKERBY MINE FACILITIES

PREPARED FOR:
FALCONBRIDGE LIMITED

TROW ONTARIO LTD.
Toronto, Hamilton, London,
Sudbury, North Bay, Ottawa

Project: S861056M
September 12, 1986

1074 Webbwood Drive
Sudbury, Ontario P3C 3B7
(705) 674-9681

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CRAELIUS DRILL DATA Appendix 'A'

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CORRECTED PRESSUREMETER CURVES Appendix 'C'

## 1.0 TERMS OF REFERENCE

It is understood that Falconbridge Limited has undertaken a government research contract related to "in-situ modelling of a cemented sill mat and confines during tertiary stage pillar recovery."

Subsequent to being awarded this contract, Falconbridge Limited retained the professional services of Trow Ontario Ltd. for geotechnical input related to hydraulically placed tailings mine fill with the following general terms of reference:

(i) Propose and apply various sampling and drilling techniques related to mine fill core recovery towards producing a suitably precisioned hole for instrumentation purposes.

(ii) Secure and log recovered samples for laboratory analyses.

(iii) Provide in-situ instrumentation of boreholes to assess parameters of the mine fill.

(iv) Undertake a variety of laboratory tests to determine chemical and physical properties.

(v) Prepare a factual report of our findings related to the on-site drilling, core recovery, in-hole instrumentation and laboratory testing. Additionally, Trow was requested to provide comments and recommendations related to future mine fill drilling and testing.

## 2.0 SCOPE OF WORK

### 2.1 Research Location (Lockerby Mine Complex)

Considering the logistical problems related to the research drilling and instrumentation (i.e., in combination with timing and mine production schedules), it was recommended by Trow Ontario Ltd. that the field drilling be located directly into the exposed mine fill face.

The obvious advantages to direct mine fill access were to eliminate the time required for rock collaring and to provide optimum visual control and evaluation of both the drilling procedures and integrity of the boreholes.

The area selected by Falconbridge Limited for the research drilling was located at the 2950 level of the Lockerby Mine Complex within the 31-176 stope O/C. This, in turn, provided access to stopes 31-172 and 31-179 which had been previously backfilled in 1982 and 1985, respectively. Plan details are shown on Figure 1 on page 3.

## 2.2 Classified Mill Tailings:

### 2.1.1 General:

As noted, filled stopes 31-172 and 31-179 were backfilled in the years 1982 and 1985, respectively, with hydraulically placed, classified mill tailings. It is understood that portions of the fill may contain up to 20% glacial sand added and all tailings are mixed with various percentages of Portland cement.

Typical grain size distribution curves of the backfill, as supplied by Falconbridge Limited, are shown on the enclosed Figure 2, on page 4. with an average distribution curve plotted on Figure 3 on page 5.

As can be seen, particle distribution sizes range on an average from 0.035 mm to 0.60 mm in diameter which, in geotechnical terms, equates to a fine to medium silty sand.

With the addition of cement, relationships between compressive strength and cement content have been developed by Falconbridge Limited and are illustrated on Figures 4 and 5, pages 6 and 7, respectively.

Classified mill tailings are processed and transported from Falconbridge's Strathcona Mill, Onaping Falls, Ontario.

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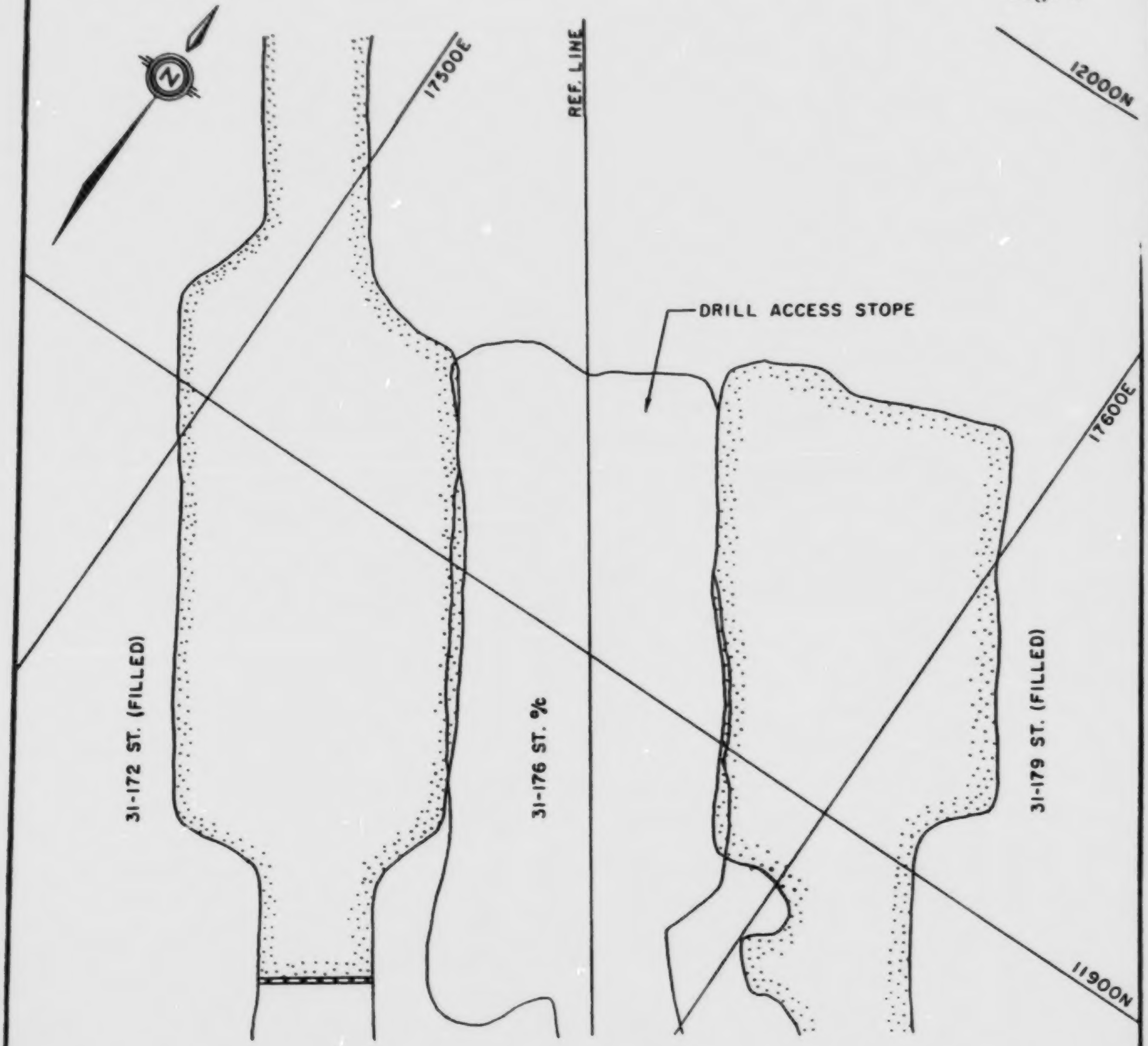
Fig. 1


17500E
REF. LINE
12000N
DRILL ACCESS STOPE
17600E
31-172 ST. (FILLED)
31-176 ST. %
31-179 ST. (FILLED)
11900N
N

PLAN

Not To Scale

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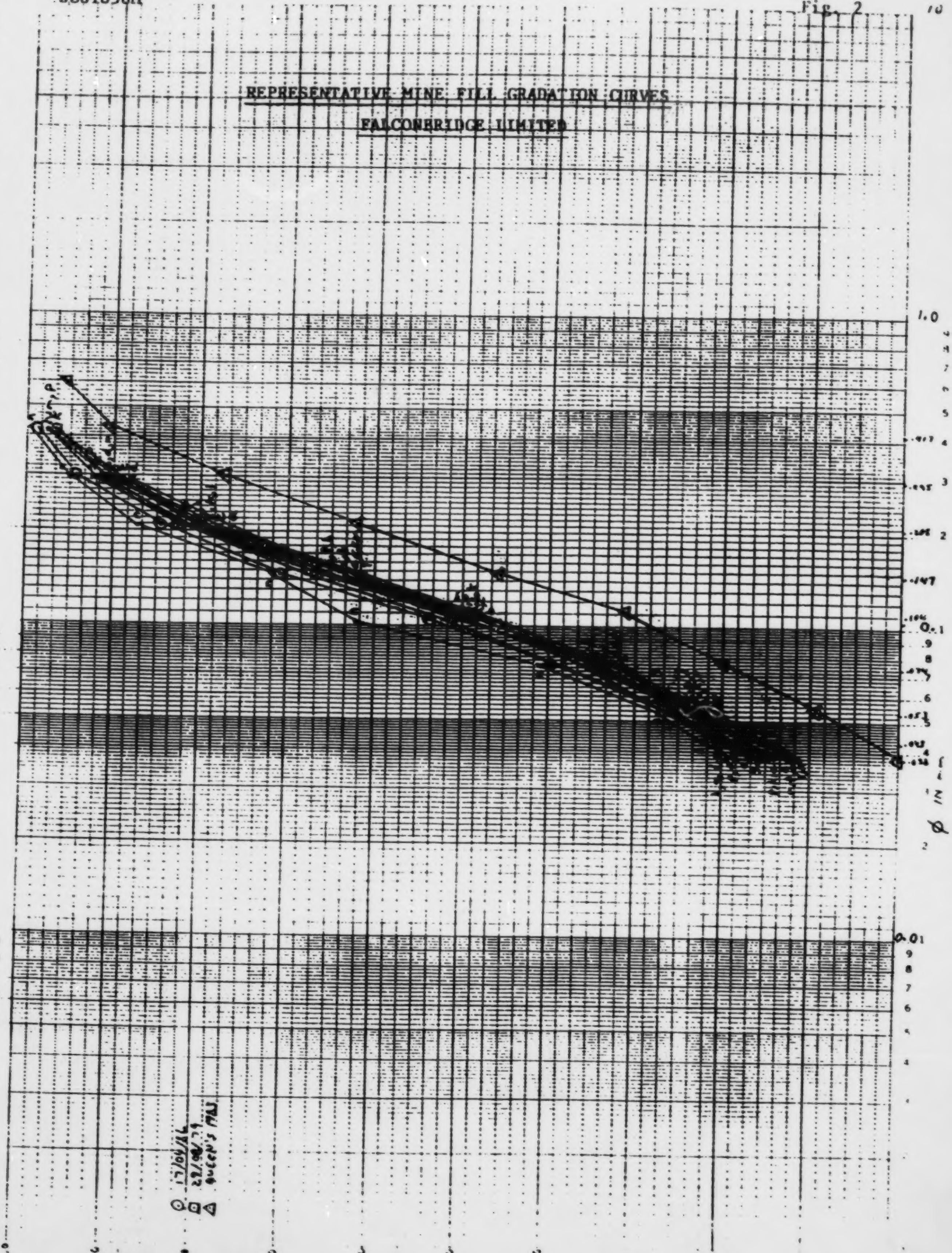
TROW ONTARIO LIMITED

S861056M

Fig. 2

# REPRESENTATIVE MINE FILL GRADATION CURVES

## FALCONBRIDGE LIMITED

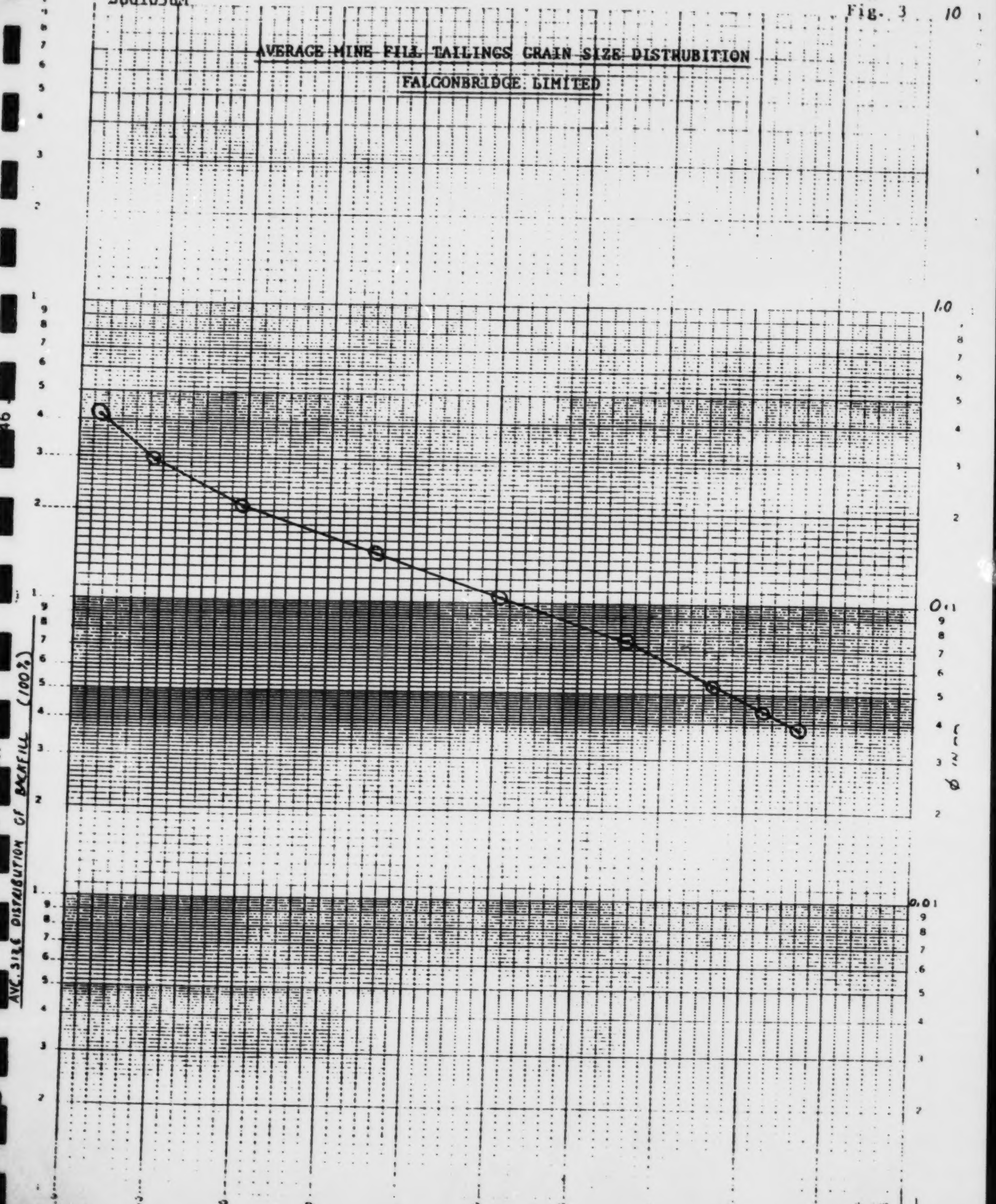


S861056M

Fig. 3

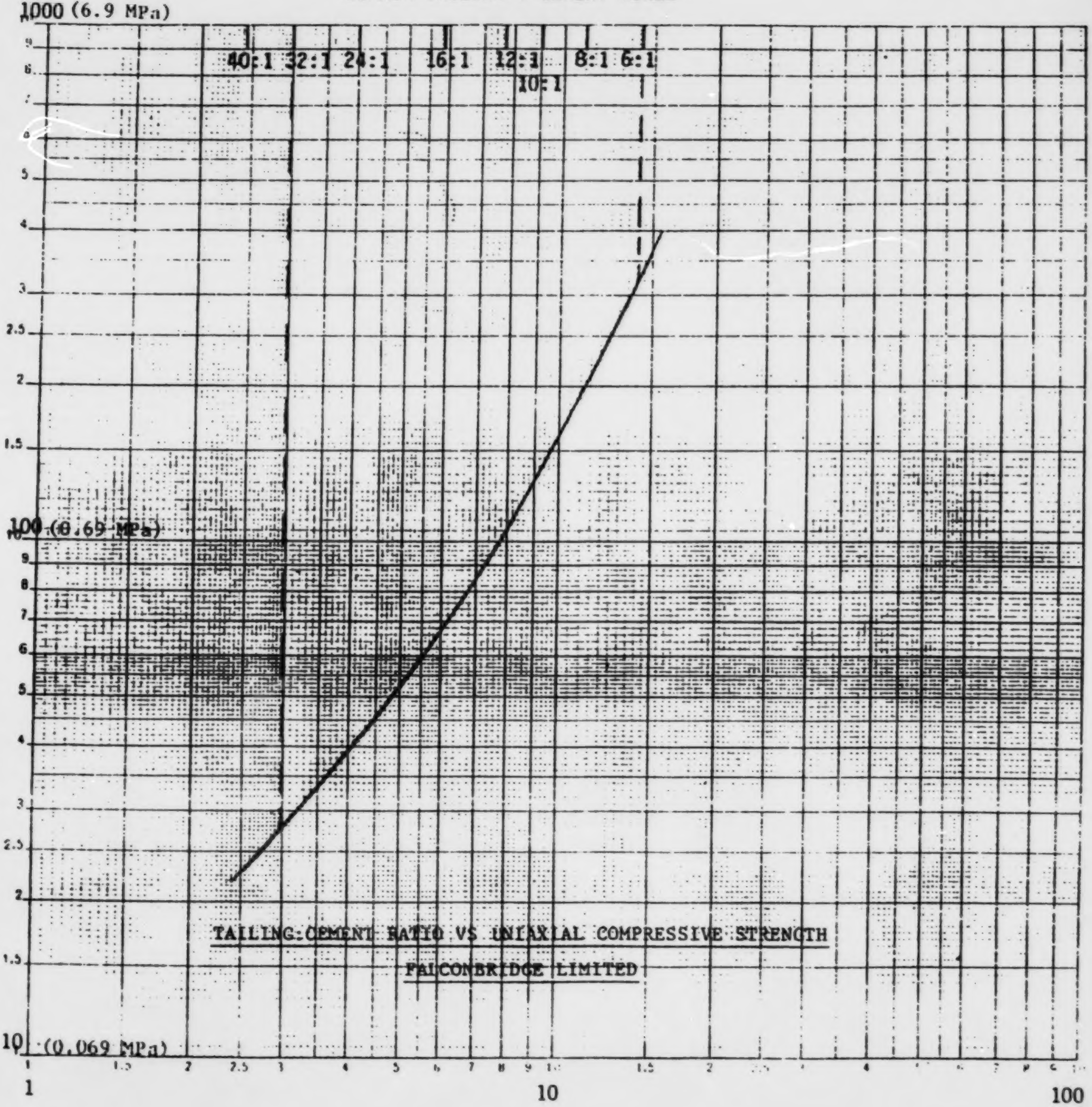
## AVERAGE MINE FILL TAILINGS GRAIN SIZE DISTRUBITION

## FALCONBRIDGE LIMITED



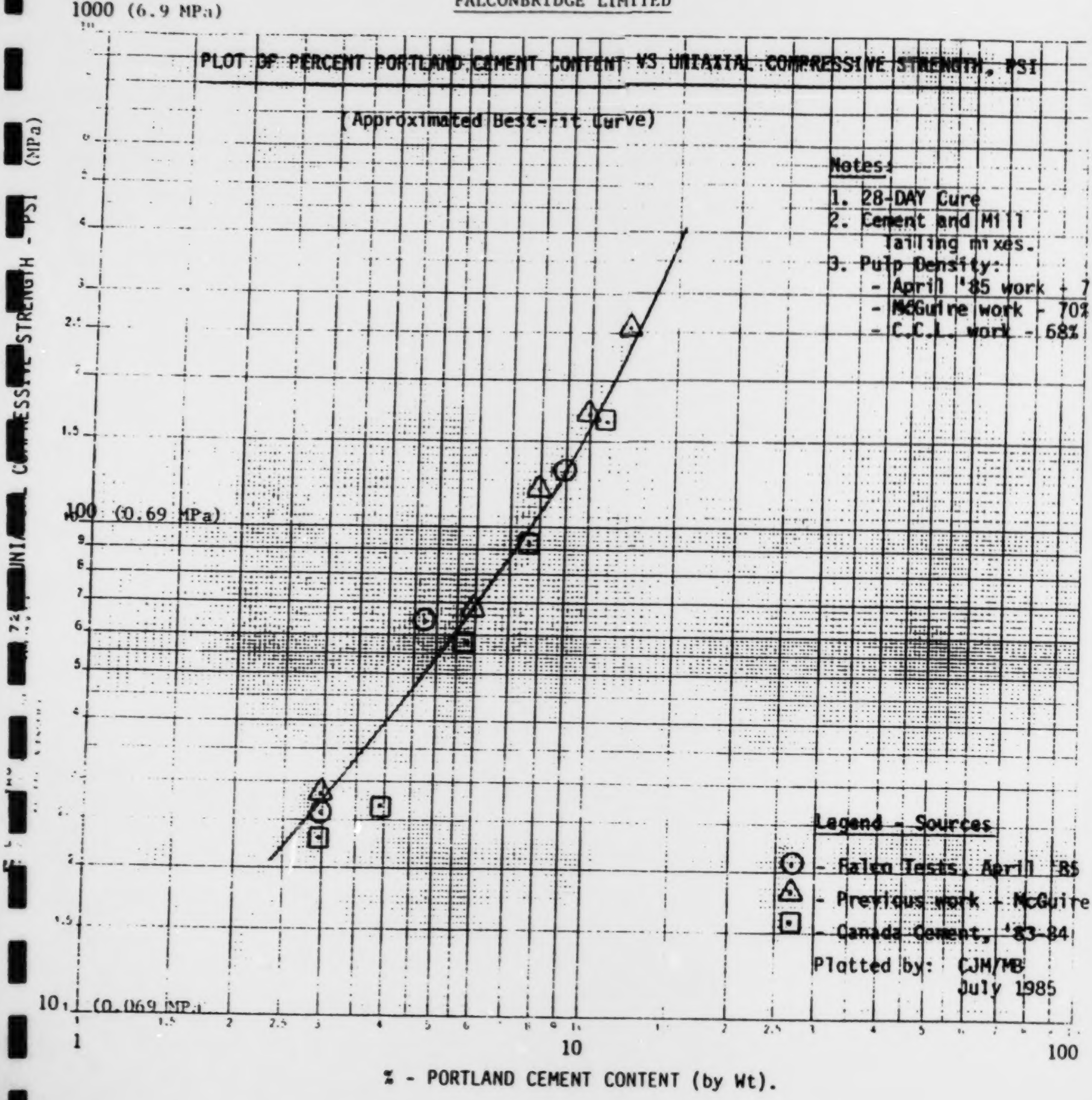
# EMPIRICAL RELATIONSHIP
# 28-DAY COMPRESSIVE STRENGTH VS
# PERCENT PORTLAND CEMENT CONTENT

COMMON TAILING : CEMENT MIXES



% - PORTLAND CEMENT CONTENT (by Wt.)

## % PORTLAND CEMENT BY WEIGHT
## VS UNIAXIAL COMPRESSIVE STRENGTH
## FALCONBRIDGE LIMITED



### 2.2.2 Observations of Mine Fill Face:

As noted previously, drilling was accessed from stope 31-176 O/C with a base elevation of approx. 10,097*. Adjacent filled stopes were generally in the order of 11 m wide by 21 m to 24 m long. It is understood that fill heights above the access base elevation were in the order of 116 m.

It is understood that the design tailings:cement ratio for the 31-172 stope test area was 12:1 with a 16:1 ratio for the 31-179 stope test area.

Visual examination of the exposed faces of the mine fill display varying colours and stratifications which are typical of hydraulically placed mine fill. Variations have resulted from periodic infilling schedules, discharge segregation, variable particle settlements, etc.

Two representative fill exposure sections were selected from stope 31-172 and are illustrated on photographs 1 and 2, page 9. These particular sections were measured and detailed, and each major layer and stratification was sampled. The recovered samples were then submitted to Falconbridge Limited for cement content determination. These results are summarized on Figures 6 and 7 on pages 10 and 11, which highlight the variability of cement/tailings content within the mine fill.

While no detailed measurements of the inclination of the major stratifications were made, general observations indicated that the layering is generally at an angle of about 5° to the horizontal.

---

**Falconbridge Limited datum*

Photograph #1

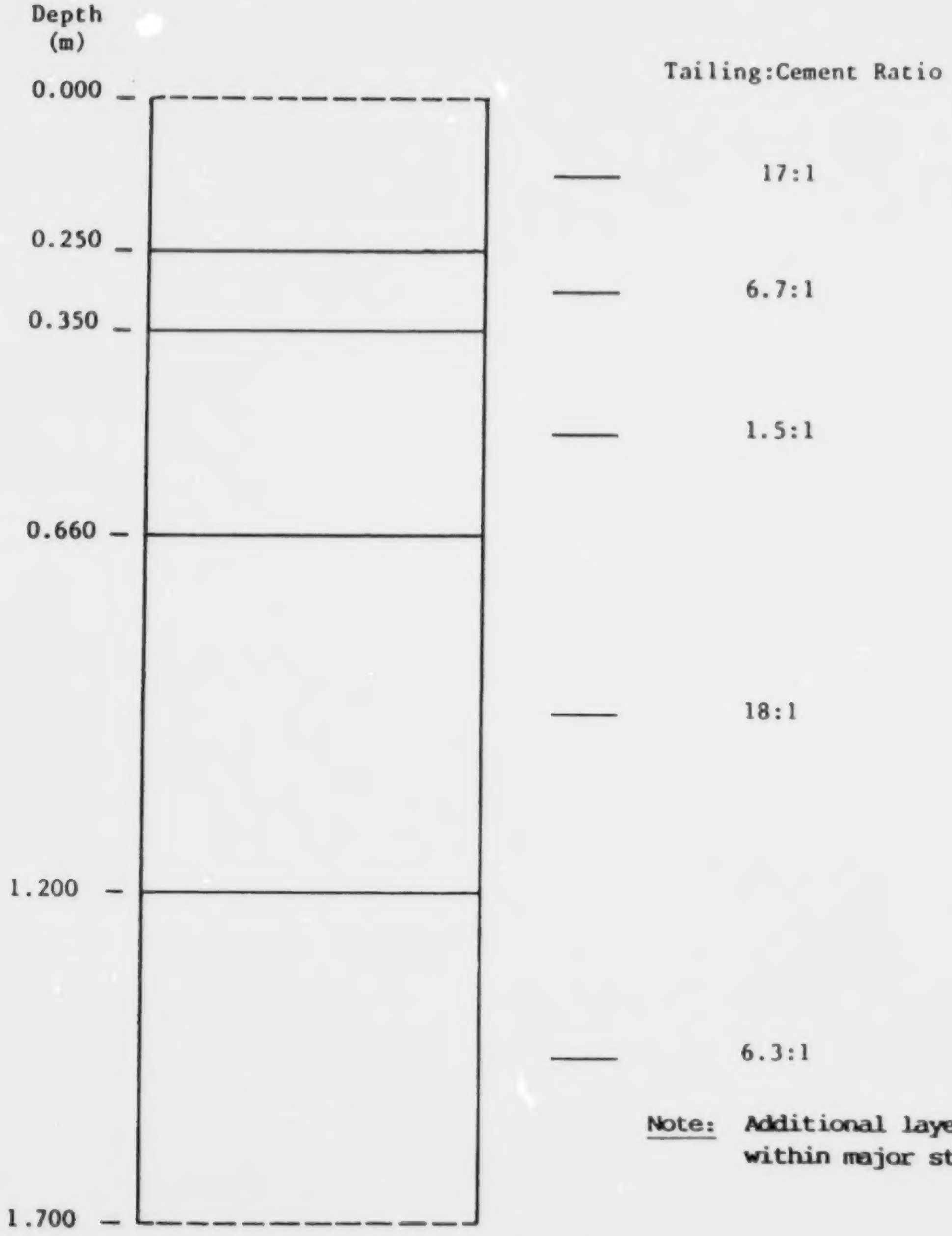
In-place mine fill stratification - centre of stope 31-172 (see Fig. 6 on page 10)

Photograph #2

In-place mine fill stratification - northerly end of stope 31-172 (see Fig. 7 on page 11)

## Section 1-1

## Showing Major Mine Fill Stratifications and Tailing:Cement Ratio


Depth
(m)
0.000
0.250
0.350
0.660
1.200
1.700
Tailing:Cement Ratio
17:1
6.7:1
1.5:1
18:1
6.3:1

Note: Additional layering evident within major stratifications

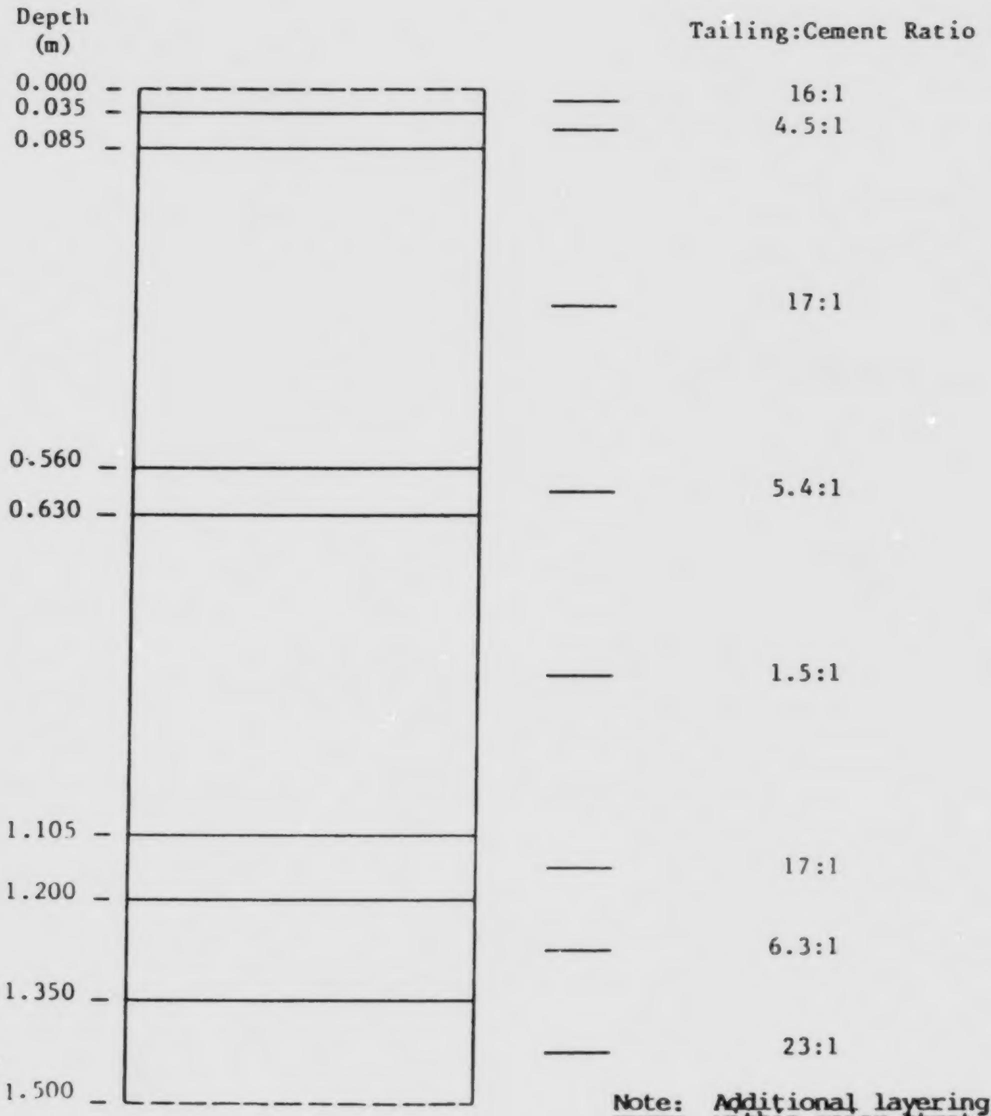
CENTRE OF STOPE 31-172

Fig. 7

Section 1-2

Showing Major Mine Fill Stratification

and Tailing:Cement Ratio


Depth (m)
Tailing:Cement Ratio
0.000
0.035
0.085
0.560
0.630
1.105
1.200
1.350
1.500
16:1
4.5:1
17:1
5.4:1
1.5:1
17:1
6.3:1
23:1

**Note:** Additional layering evident within major stratifications

NORTH SIDE OF STOPE 31-172

## 2.3 Borehole Locations:

A total of six boreholes were advanced into the mine fill, i.e., boreholes 1 to 4 drilled into stope 31-172 and boreholes 5 and 6 drilled into stope 31-179.

Alignment and borehole data are recorded as follows:

| BOREHOLE NO. | HOLE DIAMETER (mm) | BOREHOLE ANGLE | COLLAR ELEVATION* | BOREHOLE DEPTH (m) |
|---|---|---|---|---|
| 1 | 95 | 0.0° | 10,101.3 | 5.8 |
| 2 | 90 | +13.5° | 10,103.2 | 5.9 |
| 3 | 90 | + 5.0° | 10,100.7 | 8.5 |
| 4 | 90 | +46.0° | 10,108.3 | 7.4 |
| 5 | 90 | +45.0° | 10,108.6 | 8.0 |
| 6 | 90 | +11.0° | 10,100.9 | 7.5 |

Borehole plan and section locations are graphically detailed on Figures 8 and 9 on pages 13 and 14.

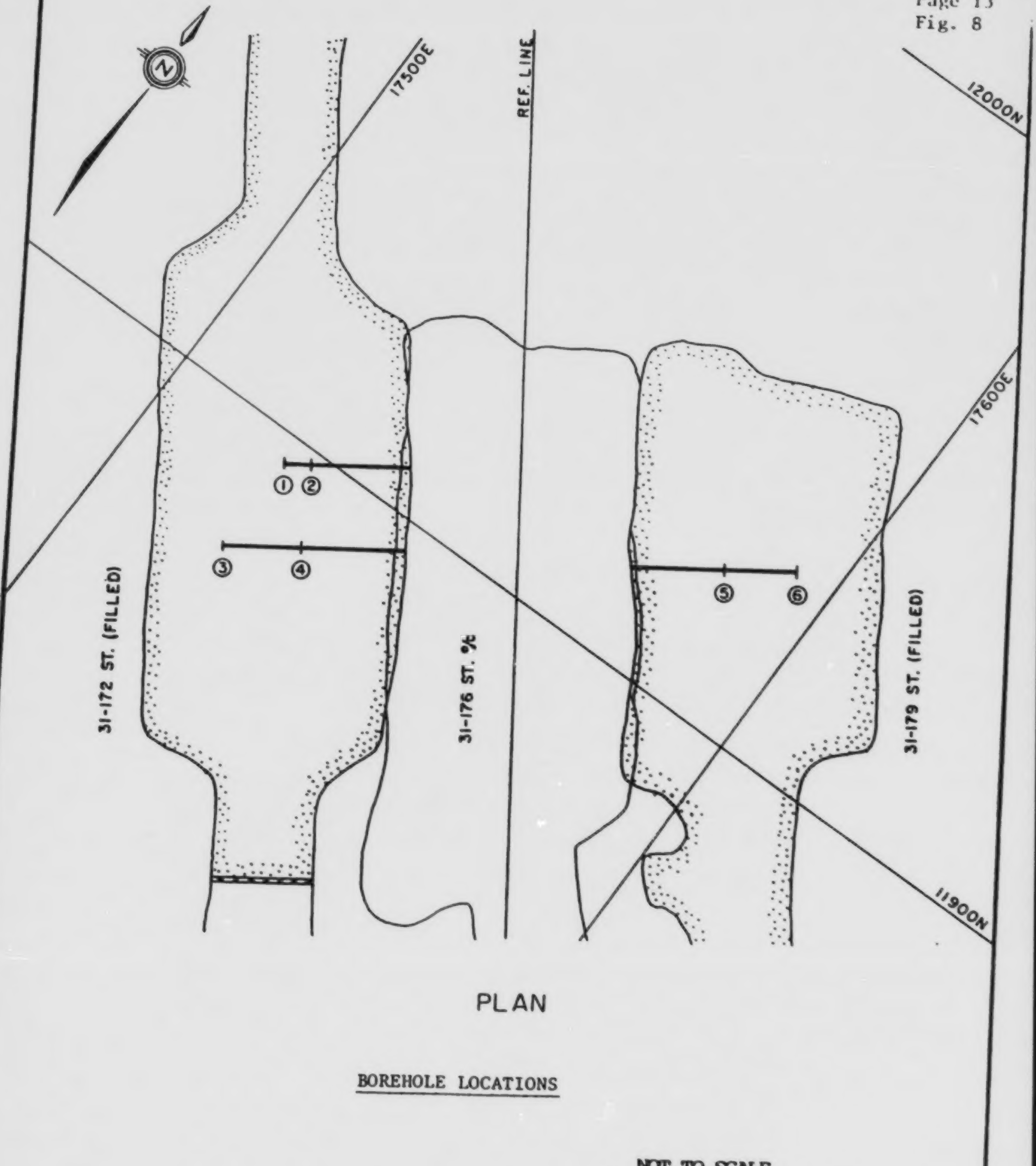
Borehole 1 was used for trial drilling purposes only and was not used for sampling or instrumentation purposes.

---

*Falconbridge Limited datum

PROJ. Nº S861056M

Fig. 8


N
17500E
REF. LINE
12000N
17600E
11900N
31-172 ST. (FILLED)
31-176 ST. %
31-179 ST. (FILLED)
① ②
③ ④
⑤ ⑥

PLAN

BOREHOLE LOCATIONS

NOT TO SCALE

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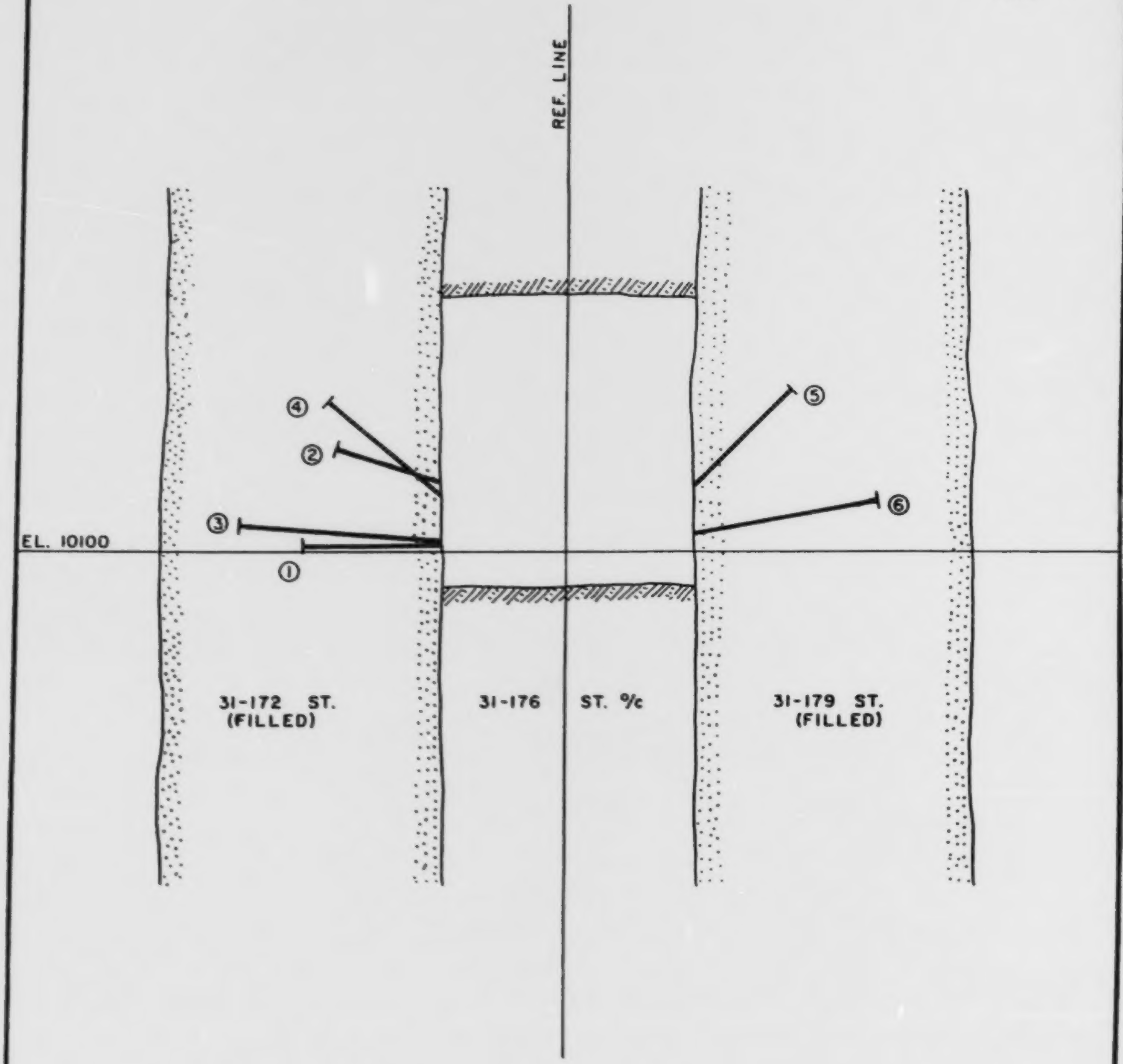
REF. LINE
EL. 10100
①
②
③
④
⑤
⑥
31-172 ST.
(FILLED)
31-176
ST. %c
31-179 ST.
(FILLED)

SECTION

BOREHOLE LOCATIONS

NOT TO SCALE

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## 2.4 Drilling Equipment:

Based on past recommendations and requirements for a more geotechnical oriented drilling application, a significant advance related to in-situ drilling and sampling of mine fill was made when Falconbridge Limited commissioned the use of a variable hydraulic drill system for this research program.

The drilling was undertaken using a skid-mounted Craelius DIAMEC ~~dynamic~~ 251 hydraulic drill unit. The feed mechanism consisted of a chain advanced with hydraulic cylinders. This, in turn, advanced a rod rotation unit operating in the range of 25 rpm to 600 rpm (i.e., low speed rotation unit). The unit also carried a hydraulic N-size rod holder and was powered by a Craelius Model 30E electric power unit with a motor rating of 30 kW. Also supplied by Craelius was a T65 86 core barrel which was designed with an internal separating (split) core container.

Specific data related to the Craelius drill unit is contained in Appendix A at the end of this report.

Additional ancillary in-hole and sampling equipment was supplied by the Diamond Drilling Division of Falconbridge Limited; Trow Ontario Ltd., and DBM Technical Drilling.

## 2.5 "Flushing" Agents & Methods:

For the purpose of borehole advancement and subsequent removal of mine fill cuttings, the following methods were evaluated as part of this project:

Co-Polymer mixed with pressurized water
Pressurized water
Pressurized air

The co-polymer used was a concentrated, granular co-polymer of sodium acrylamides/acrylates supplied by Westcoast Drilling Supplies Ltd.

## 2.6 Instrumentation:

In order to measure the in-situ properties of the mine fill, two separate methods of instrumentation were proposed for use on this project.

Primarily, the use of a model Texam pressuremeter, owned by Trow Ontario Ltd. was utilized on this research program for the purpose of establishing stress-strain characteristics and modulus of deformation values.

The Texam pressuremeter was used in conjunction with an NX size probe with a flexible, rubberized, expanding sheath.

Features and applications of the Texam pressuremeter (in brief) are included in Appendix 'B'.

Additionally, the use of a static cone penetrometer apparatus supplied by Roctest Ltd. was attempted with the intent of evaluating penetration resistance.

## 2.7 Laboratory Testing:

As part of the evaluation of recovered mine fill core samples, the following laboratory testing was performed:

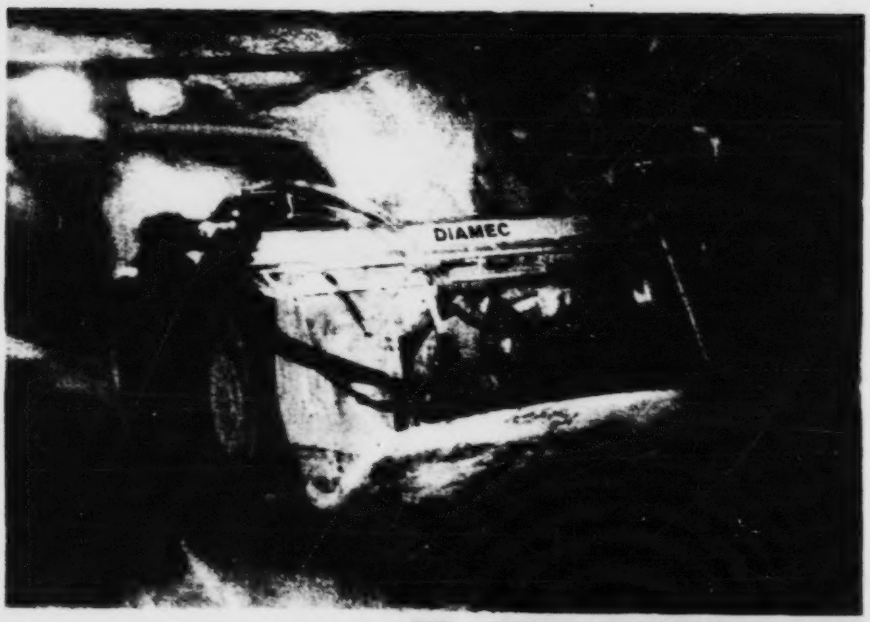
(i) Moisture contents
(ii) Unit weights
(iii) Unconfined uniaxial compressive strengths
(iv) Triaxial compressive strengths
(v) Cement content determination (carried out independently by Falconbridge Limited.)

# 3.0 DRILLING - METHODS & TECHNIQUES

## 3.1 Access & Set-up:

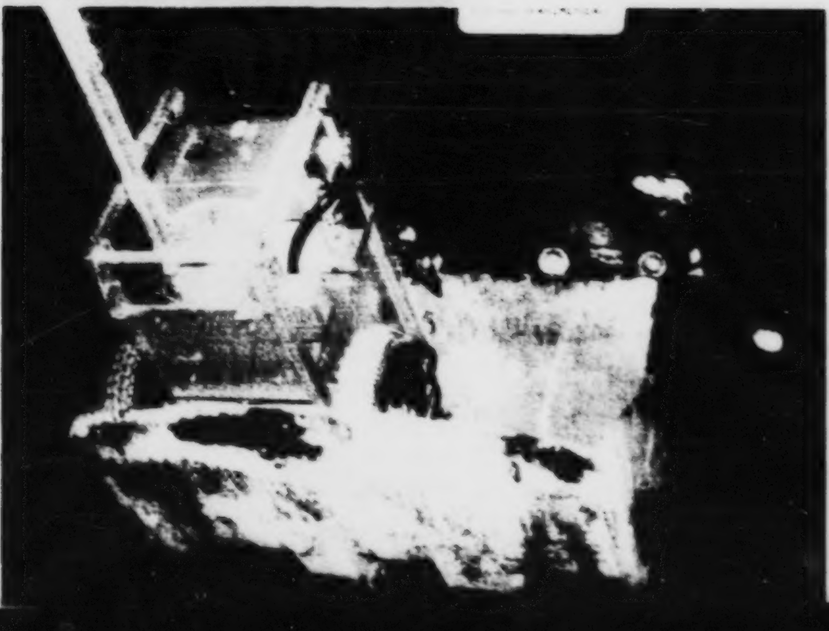
Since the feed frame and power unit, etc., of the Craelius drill are separate components, access and transportation, both from surface and within the test stope area, presented no major problems.

Photographs 3 and 4 show the use of a "scoop tram" to load and transport the various drilling components.



Photograph #3

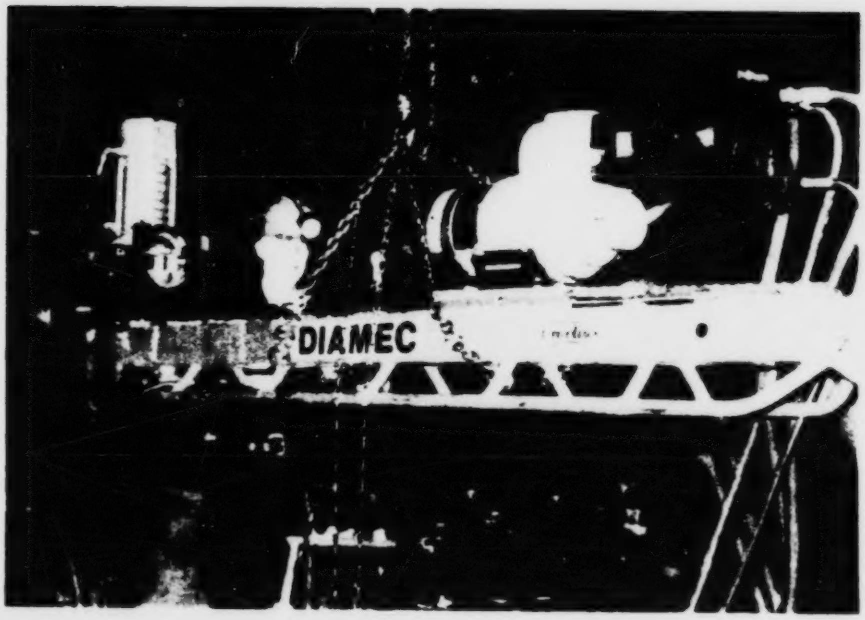
Craelius 251 feed frame being transported underground



Photograph #4

Craelius Model 30E power unit being transported underground

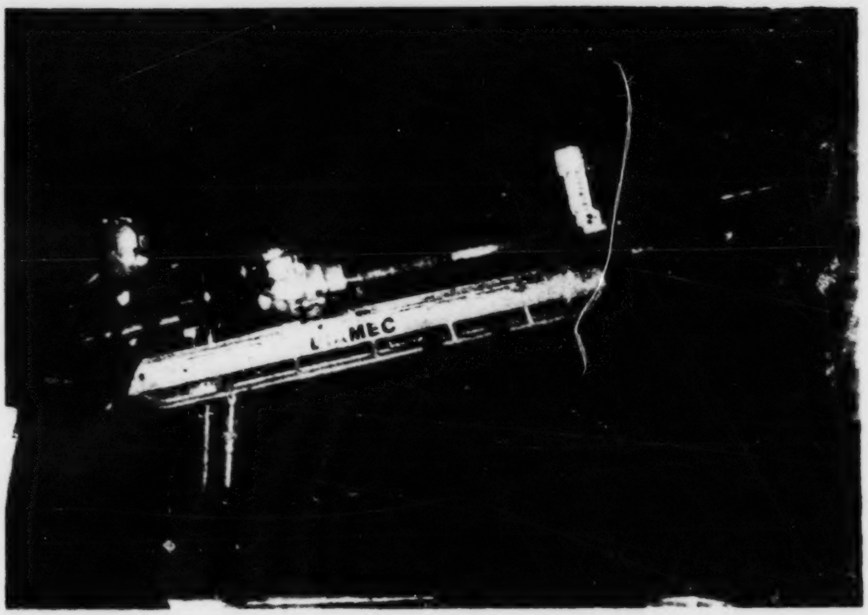
Assembly and alignment of the feed frame, utilizing overhead block and tackle apparatus, is shown in photograph 5.



Photograph #5

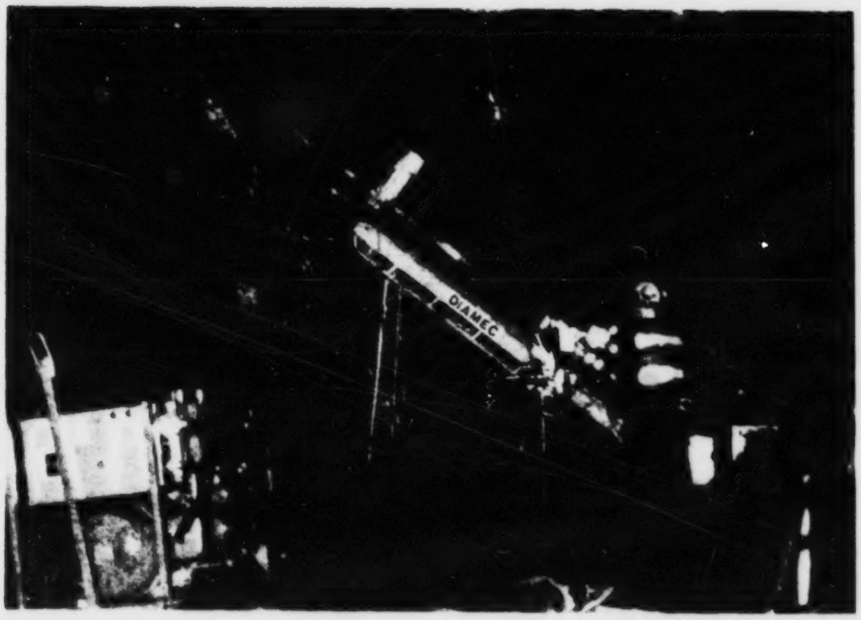
Assembly of Drill Feed Frame

A typical set-up for actual drilling operations is illustrated in photographs 6 and 7.



Photograph #6

Typical set-up & drillir
operations



Photograph #7

Typical set-up &
drilling operations

A 550V, 50 amp service line was installed by Falconbridge Limited to power the Craelius Model 30E electric motor prior to our arrival.

Water for both drilling and in-line cooling of the electric power unit was readily available within the stope area.

## 3.2 Drill Pressures, Feed Rates, RPM Data:

Based on initial sample recovery efforts using rotary wash boring techniques in the exploratory borehole 1, optimum ranges of drill pressures and feed rates, etc., were established and then used during drilling in subsequent boreholes.

Feed Load - 0.5 - 0.75 tonne
Water Pressure - 200 to 400 kPa
Feed Rates - 15 to 25 mm/min.

The average speed of the rotation was estimated to be within the range 50 to 150 rpm's.

These values for the shallow holes advanced were found to provide optimum penetration with minimum "flushing" requirements and disturbance of the mine fill samples.

## 3.3 Sample Retrieval - Undisturbed:

### 3.3.1 Sampling Techniques:

Given the requirement for securing undisturbed samples, Trow Ontario Ltd. concentrated on the use of double tube core barrels and thin wall sampler such as industrial single tube core barrels and split barrel (thin walled) samplers. The basic objective was to secure samples, relatively intact, with a minimum of disturbance from drilling operations and "flushing" agents.

### 3.3.2 Thin Walled Split Barrel Sampler:

This method incorporated the use of a standard 75 mm thin wall sampler (shelby type) which had been machine cut to provide a split-barrel concept for easier sample removal. The cutting edge was additionally machine sharpened at a bevelled angle of approximately 45°. The sampler was then advanced into the backfill by static pressure provided by the drill unit.

Observations related to sampling attempts indicated that this type of sampler was relatively ineffective. High static pressures were required to penetrate even the weakly cemented backfill zones and resulted in damage to the sampler and drill string. Additionally, it was found that the sample (under pressure) would expand and "block" within the sampler and invariably fracture.

### 3.3.3 Thin Wall Single Tube Sampler:

For this purpose, a 100 mm diameter, industrial concrete core barrel was used and advanced with a water and co-polymer "flushing fluid" combination.

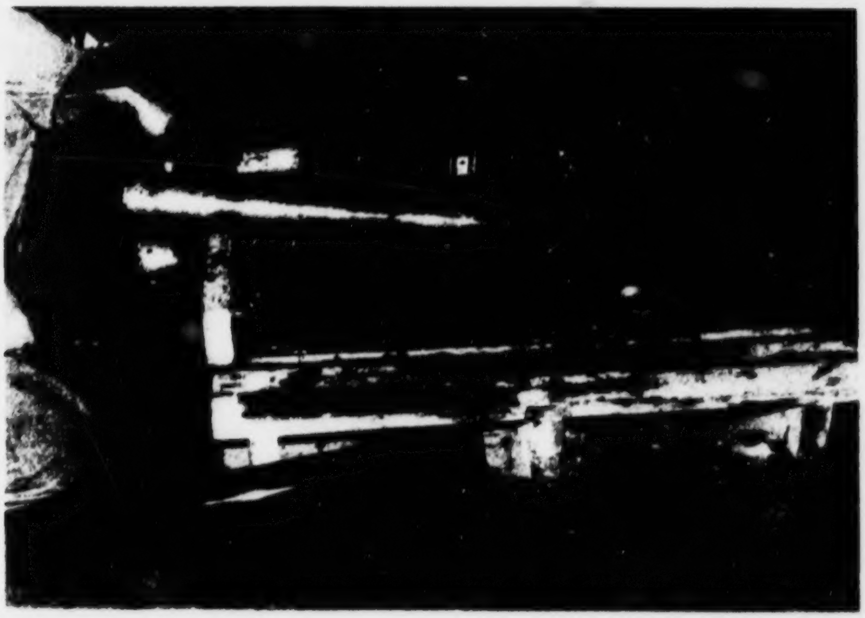
Previous experience with this type of sampler had yielded relatively good results in sample retrieval for the more highly cemented fills. However, its use is restricted to inclined or horizontal holes, as the single tube design does not include a sample retaining system for recovery of samples from "down-hole" drilling.

As an alternative on this project, the single tube sampler was advanced into a more weakly cemented stratum at stope 31-179.

Photograph 8 illustrates the core recovered on this trial run and highlights the following results:

- Core recovery was 90% (i.e., 10% remained in hole)
- Recovered sample was saturated and without exception disturbed.

Given the requirement that drilling fluid is in total contact with the sample at all times, it was obvious that the more weakly cemented mine fill core was adversely affected and badly disturbed using this sampling apparatus.



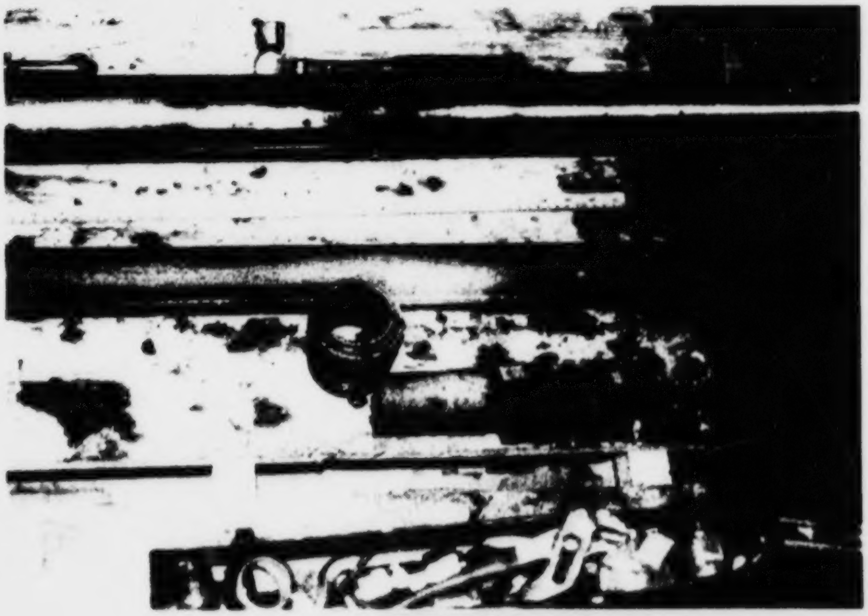
Photograph #8

Showing thin wall, single tube sampler
and recovered sample which is saturated
and quite disturbed

### 3.3.4 Double Tube Core Barrels:

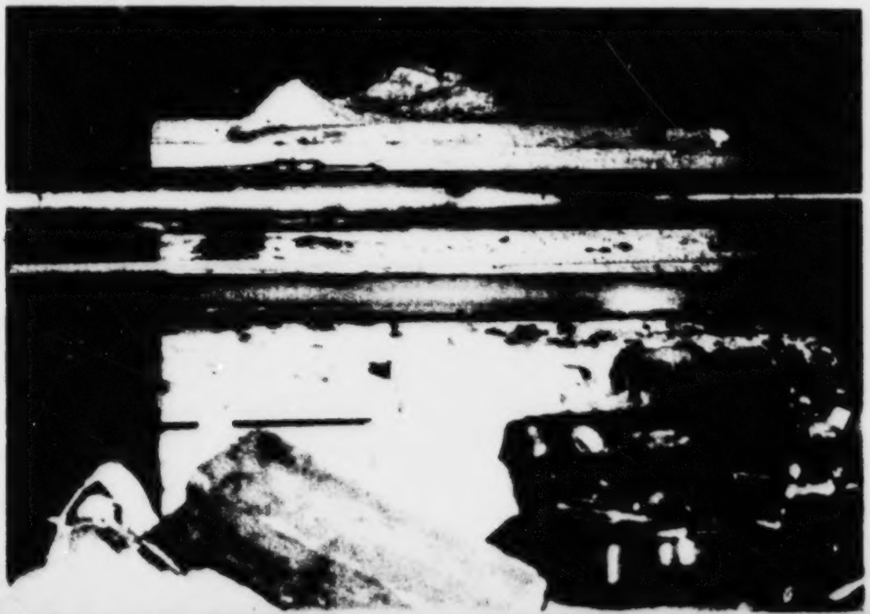
For core retrieval purposes, both a Craelius inner split tube and standard NW solid tube core barrel were used.

Subject to establishing the optimum drilling parameters (i.e., water and feed pressures, etc), core recovery in all ranges of mine fill strengths (using either type of core barrel) was in the order of 95% to 100%. Typical examples are shown on the following photographs 9, 10 and 11.



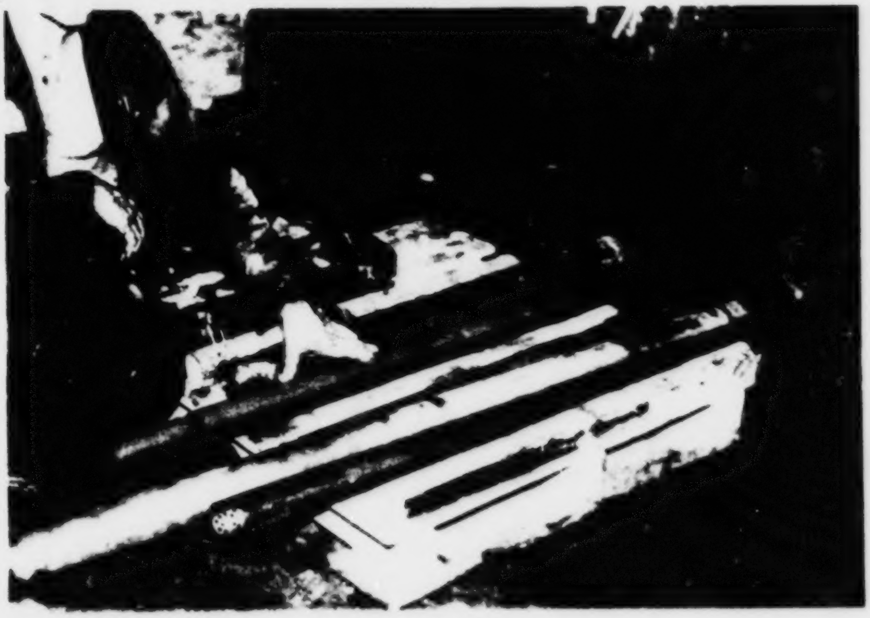
Photograph #9

Typical core recovery using Craelius split inner tube core barrel (core recovery 100%)



Photograph #10

Core recovery with Craelius core barrel (core recovery 100%)



Photograph #11

Typical core recovery using NX solid inner tube core barrel. Here, sample has been transferred from tube to sample tray (core recovery 100%)

All samples recovered were virtually undisturbed, with the exception of some degree of water saturation. However, as opposed to the single tube sampler, the double tube design concentrated any water exposure to a limited location at the drill bit and resulted in excellent core integrity. For example, "weak" samples with uniaxial strengths in the low range of 150 kPa remained sufficiently intact for triaxial laboratory testing.

Both core barrels worked equally efficiently in drilling and sample retrieval, although the split barrel design of the Craelius core barrel provided easier logging and handling of recovered samples.

Core retention rings were removed from both core barrels during initial trial runs, as they were found to cause drilling blockages in the more sensitive fill layers. Where core retrieval was in question, the last 75 to 100 mm of the core run was advanced "dry" with increased pressure to secure the sample in the core barrel.

### 3.4 "Flushing" Agents:

The co-polymer/water combination provided the most effective means of "cutting", removal and hole stabilization during coring operations. Conversely, drilling without the co-polymer required increased water, which subsequently aggravated the potential for saturation and disturbance of the core samples.

When used under carefully controlled conditions, neither the water nor co-polymer/water caused excessive distortion of the borehole walls.

Visual examination of the in-hole condition after drilling, together with a series of volume measurements (using the pressuremeter probe), indicated relatively uniform conditions with borehole wall deformation maintained within a tolerance of approximately 6.5 mm.

Air used on several occasions was found to present a safety hazard (i.e., dust, pressurized sand particles, etc.). Subject to a filter or buffer system being designed at the collar location, its use is considered restrictive at this time.

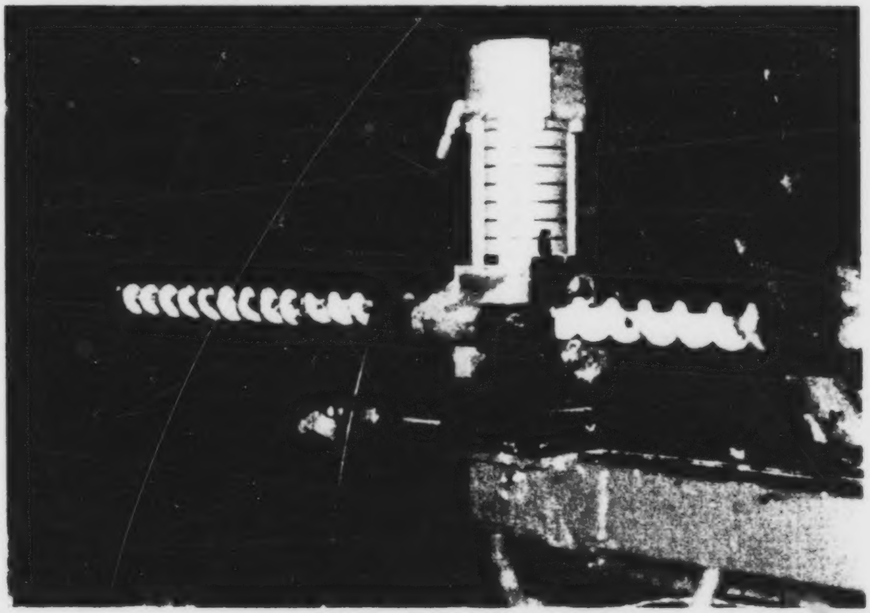
### 3.5 Continuous Flight Augers:

For general access drilling, disturbed sampling, and precision hole sizing for potential instrumentation, the use of continuous flight augers was found to perform extremely well.

This method, without having to use air or water, was found to be an excellent economical and speedy means of securing disturbed samples for cement content or moisture content determinations.

Additionally, the hole, properly drilled and sized, could provide excellent tolerances for instrumentation purposes, i.e., pressuremeter, extensometers, etc.

Typical drilling operations utilizing helical augers are shown on photographs 12 and 13.

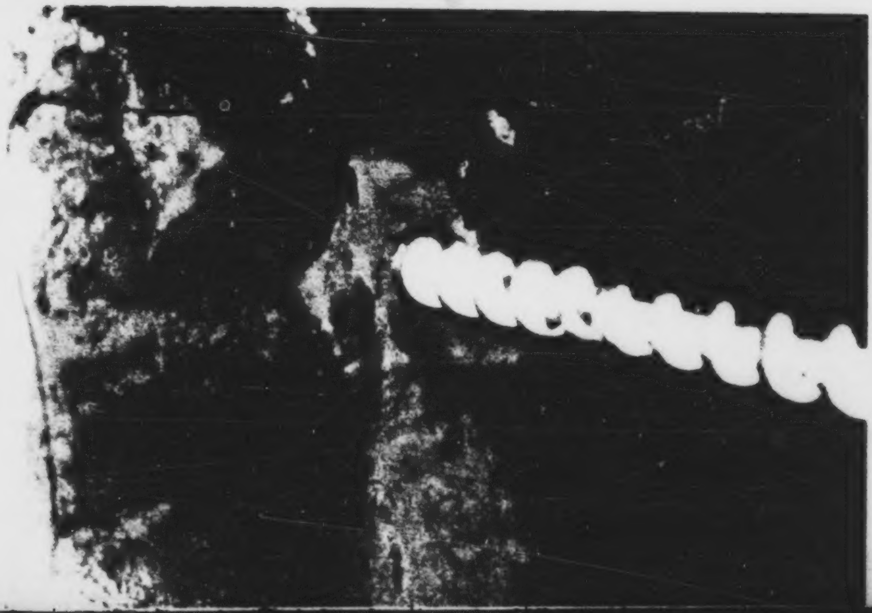


Photograph #12

Showing solid stem augers advancing through rod holder into mine fill

Photograph #13

Showing "cuttings" being removed by helical auger operations



## 4.0 SAMPLE RECOVERY AND HANDLING

We have included this section on sample handling, as it becomes an important variable when undisturbed laboratory tests are required on weak mine fill samples.

Considering the sensitive nature of the more weakly cemented fill zones (i.e., uniaxial strengths less than 200 kPa), the samples must be packaged and transported with extreme care.

All samples, once protected from moisture loss, should be packaged and installed in covered core boxes, then transported to surface by hand. Core boxes should never be carried at an angle. Vibration of core samples must be avoided.

## 5.0 INSTRUMENTATION

### 5.1 Pressuremeter:

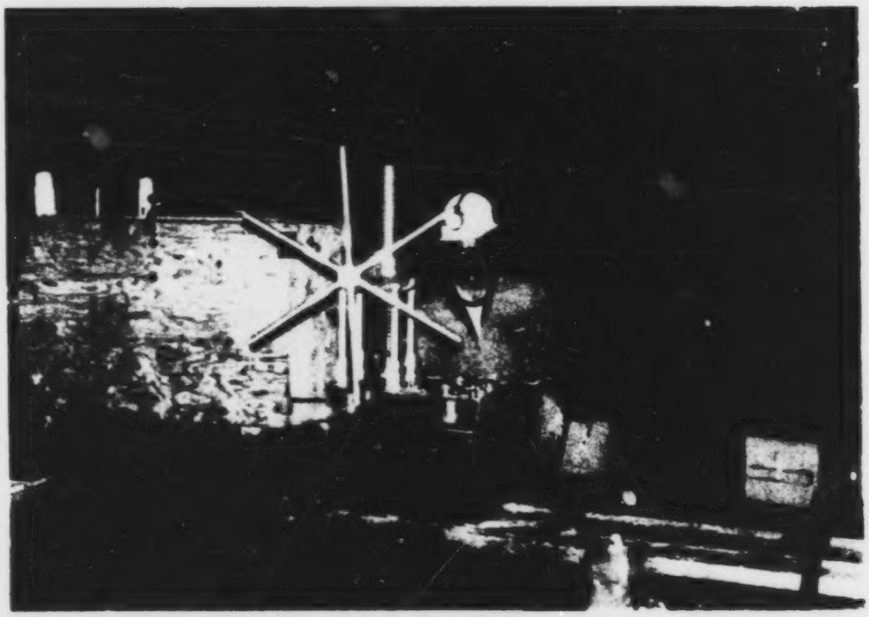
A total of thirty-one pressuremeter tests were performed on this project at varying depths within boreholes 2 to 6, inclusive, i.e., borehole 1 was used for trial drilling purposes only.

A typical installation of the NX size pressure probe is shown on photograph 14 on page 28, while the set-up for the Texam control base is illustrated in photograph 15.

A summary of pressuremeter results is shown in Table 1 on page 29, while individual stress-strain curves are detailed in Appendix C.

Photograph #14

Showing installation of NX
pressuremeter pipe into
borehole 4



Photograph #15

Showing Texam pressuremeter
control base during testing

## TABLE 1

## PRESSUREMETER TEST RESULTS

| BOREHOLE NO. | TEST NO. | DEPTH TO CENTRE OF PROBE (m) | TYPE OF TEST CONTROL | Vo (cc) | Po (kPa) | Pℓ (MPa) | $E_m$ (MPa) | REMARKS |
|---|---|---|---|---|---|---|---|---|
| 2 | 1 | 5.49 | strain | 388 | 550 | 13.0 | 235 | |
| | 2 | 4.88 | stress | 472 | 550 | 13.0 | 531 | |
| 3 | 1 | 3.51 | strain | 595 | 150 | 3.58 | 227 | |
| | 1A | 2.59 | strain | 478 | 275 | 4.48 | 299 | |
| | 2 | 5.18 | strain | 477 | 320 | 5.18 | 167 | |
| | 3 | 6.71 | stress | 525 | 600 | 12.7 | 372 | |
| | 4 | 8.23 | stress | 487 | 600 | 15.7 | 467 | |
| 4 | 1 | 5.49 | stress | 968 | 300 | -- | 70.9 | |
| | 2 | 6.25 | stress | 1018 | 450 | — | 77.7 | |
| | 3 | 5.33 | stress | — | -- | — | -- | line was kinked |
| | 4 | 4.27 | stress | 819 | 950 | 16.1 | 569 | |
| | 5 | 3.35 | stress | 1056 | 500 | -- | 86.4 | |
| | 6 | 2.44 | stress | — | -- | -- | — | hole too large |
| 5 | 1 | 1.37 | stress | 476 | 240 | -- | 12.6 | |
| | 2 | 0.46 | stress | 648 | 280 | — | 44.0 | |
| | 3 | 2.90 | stress | 415 | 300 | -- | 107 | |
| | 4 | 1.98 | stress | 525 | 160 | -- | 75.4 | |
| | 5 | 4.42 | stress | 359 | 260 | -- | 70.4 | |
| | 6 | 3.51 | stress | 419 | 350 | -- | 106 | |
| | 7 | 5.94 | stress | 348 | 110 | — | 68.1 | |
| | 8 | 5.03 | stress | 520 | 390 | -- | 60.0 | |
| | 9 | 7.47 | stress | 320 | 150 | 2.82 | 59.7 | |
| | 10 | 6.55 | stress | 431 | 112 | 2.75 | 40.8 | |
| 6 | 1 | 2.59 | stress | 596 | 314 | -- | 65.8 | |
| | 2 | 1.98 | stress | 738 | 450 | -- | 90.2 | |
| | 3 | 3.96 | stress | 363 | 285 | -- | 129 | |
| | 4 | 3.05 | stress | 498 | 600 | -- | 168 | |
| | 5 | 5.64 | stress | 520 | 460 | -- | 122 | |
| | 6 | 4.72 | stress | 754 | 440 | -- | 94.4 | |
| | 7 | 7.32 | stress | 478 | 520 | -- | 77.0 | |
| | 8 | 6.40 | stress | 643 | 294 | -- | 69.6 | |

Note: Pℓ = limit pressure
$E_m$= pressuremeter modulus
Vo,Po = volume at the beginning of the elastic portion of the curve at the pressure po

Some minor difficulties were experienced in providing an acceptable hole diameter for the NX pressuremeter probe. For instance, the hole produced by the Craelius core barrel was too large for any meaningful pressuremeter applications. Alternatively, an NX core barrel was used, which yielded more acceptable borehole size tolerances for pressuremeter testing purposes.

### 5.2 Static Cone Penetrometer:

Given the relatively high strength zones within the mine fill test areas (i.e., uniaxial strengths of up to 4000 kPa), in combination with the lightweight drill unit used on this project, the advancement of the static pressure core penetrometer into this particular fill was not practical.

Static penetrometer loads applied to the fill with the Craelius equipment chained to the rock floor caused the feed frame to shift and bend several drill rods; consequently, further attempts were discontinued.

## 6.0 LABORATORY TESTS

### 6.1 Moisture Content and Unit Weight Values:

The moisture contents and unit weights were determined at numerous selected locations of recovered samples of the fill. These results are summarized in the following Table 2.

## TABLE 2

## MOISTURE CONTENTS & UNIT WEIGHTS

| BOREHOLE NO. | DEPTH (m) | AVERAGE MOISTURE CONTENT* % OF DRY WEIGHT | AVERAGE UNIT WEIGHT* kg/m³ (wet weight) |
|---|---|---|---|
| 2 | 4.45 - 4.63 | 17.2% | 2178 |
| 2 | 4.69 - 4.88 | 17.3% | 2346 |
| 2 | 4.94 - 5.12 | 28.8% | 2130 |
| 2 | 5.18 - 5.40 | 18.8% | 2226 |
| 2 | 5.40 - 5.58 | 16.3% | 2219 |
| 2 | 5.70 - 5.91 | 18.0% | 2092 |
| 3 | 3.20 - 3.81 | 27.2% | ---- |
| 3 | 4.63 - 4.82 | 20.3% | 2135 |
| 3 | 4.88 - 5.06 | 21.2% | 2115 |
| 3 | 5.12 - 5.24 | 21.8% | 2149 |
| 3 | 5.88 - 5.94 | --- | 2063 |
| 3 | 6.46 - 6.65 | 22.1% | 2279 |
| 3 | 6.71 - 6.86 | 20.7% | 2206 |
| 3 | 6.92 - 7.07 | 19.4% | 2186 |
| 3 | 7.93 - 8.11 | 22.5% | 2142 |
| 3 | 8.17 - 8.29 | 24.4% | 2223 |
| 3 | 8.35 - 8.53 | 21.7% | 2167 |
| 4 | 2.20 - 2.35 | 26.3% | 2236 |
| 4 | 2.44 - 2.59 | 27.5% | 2176 |
| 4 | 2.65 - 2.80 | 29.1% | 2059 |
| 4 | 3.20 - 3.35 | 25.8% | 2058 |
| 4 | 3.41 - 3.57 | 31.7% | 2103 |
| 4 | 3.63 - 3.72 | 28.3% | 2074 |
| 4 | 4.12 - 4.30 | 22.1% | 2157 |
| 4 | 4.36 - 4.51 | 31.9% | 2001 |
| 4 | 4.57 - 4.72 | 38.3% | 1898 |
| 4 | 5.33 - 5.49 | 27.8% | 2199 |
| 4 | 5.79 - 5.82 | 24.0% | 2124 |
| 4 | 5.88 - 6.01 | 24.1% | 1789 |
| 4 | 6.07 - 6.25 | 21.7% | 2160 |
| 4 | 6.55 - 6.61 | 24.3% | 2215 |
| 4 | 6.71 - 6.86 | 24.2% | 2210 |

## TABLE 2 (Cont'd)

## MOISTURE CONTENTS & UNIT WEIGHTS

| BOREHOLE NO. | DEPTH (m) | AVERAGE MOISTURE CONTENT % OF DRY WEIGHT | AVERAGE UNIT WEIGHT* kg/m³ (wet weight) |
|---|---|---|---|
| 5 | 0.46 - 0.52 | 24.4% | 2073 |
| 5 | 1.22 - 1.28 | 27.4% | 2103 |
| 5 | 1.52 - 1.62 | 26.1% | 2137 |
| 5 | 1.68 - 2.29 | 25.8% | ---- |
| 5 | 2.59 - 3.05 | 25.0% | ---- |
| 5 | 3.51 - 3.57 | 28.6% | 1957 |
| 5 | 4.27 - 4.33 | 28.9% | 2068 |
| 5 | 4.42 - 4.60 | 23.8% | 2060 |
| 5 | 4.72 - 4.82 | 44.2% | 1925 |
| 5 | 4.88 - 5.01 | 26.2% | 2275 |
| 5 | 5.03 - 5.12 | 20.0% | 2243 |
| 5 | 5.88 - 5.94 | 23.3% | 2106 |
| 5 | 6.01 - 6.07 | 21.5% | 2002 |
| 5 | 6.10 - 6.25 | 24.6% | 1567 |
| 5 | 7.35 - 7.41 | 33.8% | 1885 |
| 6 | 1.83 - 1.92 | 28.5% | 2067 |
| 6 | 1.98 - 2.04 | 27.9% | 2178 |
| 6 | 2.44 - 2.53 | 23.7% | 2073 |
| 6 | 2.59 - 2.68 | 25.4% | 2188 |
| 6 | 3.96 - 4.05 | 29.6% | 2146 |
| 6 | 7.32 - 7.44 | 30.3% | 2358 |

**Moisture contents and unit weights based on average of two to three samples*

*Recorded moisture contents may be affected by drill fluid.*

## 6.2 Cement Contents:

Based on results supplied by Falconbridge Limited from samples secured by Trow Ontario Ltd., results of cement content determinations are detailed in the following Table 3

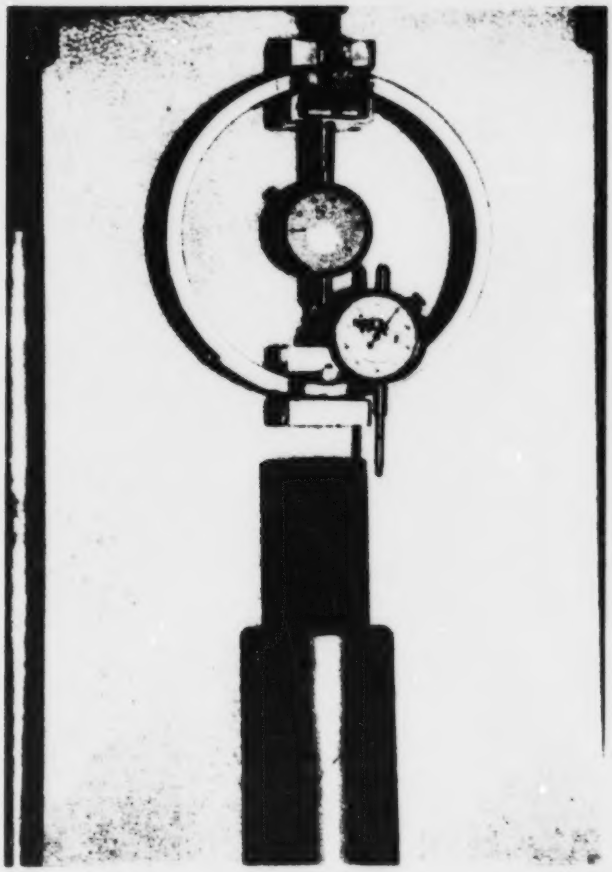
TABLE 3

SUMMARY OF CEMENT CONTENTS

| BOREHOLE NO. | DEPTH (m) | CEMENT/TAILINGS RATIO | AVERAGE RATIO |
|---|---|---|---|
| 2 | 4.88 | 12:1 | 13.5:1 |
| 2 | 5.49 | 15:1 | |
| 3 | 3.51 | 20:1 | |
| 3 | 5.18 | 13:1 | |
| 3 | 6.71 | 9:1 | |
| 3 | 8.23 | 8.8:1 | 12.7:1 |
| 4 | 2.44 | 11:1 | |
| 4 | 3.35 | 4.9:1 | |
| 4 | 4.27 | 2.1:1 | |
| 4 | 5.33 | 1.7:1 | |
| 4 | 6.25 | 17:1 | |
| 4 | 6.86 | 36:1 | 12.1:1 |
| 5 | 0.46 | 30:1 | |
| 5 | 1.37 | 27:1 | |
| 5 | 1.98 | 36:1 | |
| 5 | 2.92 | 31:1 | |
| 5 | 3.51 | 36:1 | |
| 5 | 4.42 | 33:1 | |
| 5 | 5.03 | 11:1 | |
| 5 | 5.94 | 47:1 | |
| 5 | 6.55 | 30:1 | |
| 5 | 7.47 | 21:1 | 27.5:1 |
| 6 | 1.98 | 16:1 | |
| 6 | 2.59 | 19:1 | |
| 6 | 3.05 | 40:1 | |
| 6 | 3.96 | 30:1 | |
| 6 | 4.72 | 18:1 | |
| 6 | 5.64 | 18:1 | |
| 6 | 7.32 | 19:1 | |
| 6 | 6.40 | 11:1 | 21.4:1 |

### 6.3 Uniaxial Compressive Strengths:

A typical sample prepared for uniaxial testing is shown on photograph 16, below.



Photograph #16

Uniaxial test sample being
set-up for loading

Uniaxial compressive strength tests are summarized in Table 4, below.

## TABLE 4

## UNIAXIAL COMPRESSIVE STRENGTH TESTS (kPa)

| BOREHOLE NO. | DEPTH (m) | UNIAXIAL COMPRESSIVE STRENGTHS (kPa) |
|---|---|---|
| 2 | 5.09 | 1000 |
| 2 | 5.28 | 1600 |
| 2 | 5.79 | 200 |
| 3 | 4.69 | 1200 |
| 3 | 5.91 | 200 |
| 3 | 8.00 | 1800 |
| 4 | 2.27 | 600 |
| 4 | 2.51 | 1700 |
| 4 | 2.72 | 600 |
| 4 | 3.48 | 300 |
| 4 | 3.67 | 1400 |
| 4 | 4.17 | 2400 |
| 4 | 4.26 | 500 |
| 4 | 4.43 | 2700 |
| 4 | 4.64 | 4200 |
| 4 | 5.40 | 1000 |
| 4 | 5.70 | 300 |
| 4 | 5.95 | 400 |
| 4 | 6.15 | 100 |
| 4 | 6.30 | 300 |
| 4 | 6.60 | 500 |
| 4 | 6.78 | 100 |
| 5 | 1.24 | 200 |
| 5 | 1.56 | 30 |
| 5 | 4.49 | 30 |
| 5 | 4.75 | 900 |
| 5 | 5.07 | 70 |
| 5 | 6.17 | 1100 |
| 6 | 1.82 | 30 |
| 6 | 1.92 | 10 |
| 6 | 2.01 | 10 |
| 6 | 4.00 | 100 |
| 6 | 7.38 | 1000 |

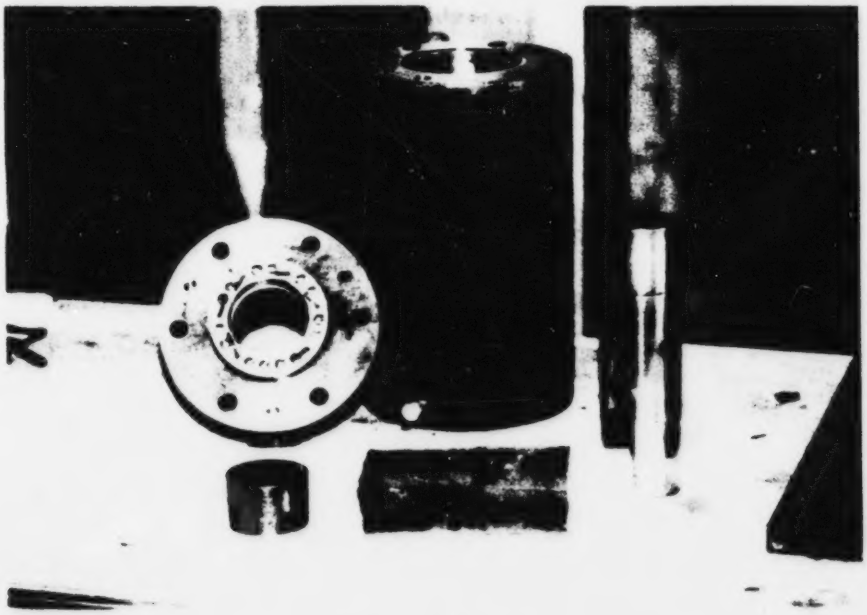
## 6.4 Confined Triaxial Compression Tests:

Confined triaxials for stress determinations were performed on selected samples of varying unconfined uniaxial strengths.

Samples carefully trimmed to a 2:1 length/diameter ratio (sample diameter approx. 87 mm) were sealed with a rubberized membrane and secured within the triaxial cell. Photograph 17, below, illustrates a prepared sample and triaxial cell apparatus. Photograph 18 on page 37 illustrates the triaxial cell prepared for principal stress loading. Photograph 19 on page 37 indicates apparatus used for confining pressure applications.

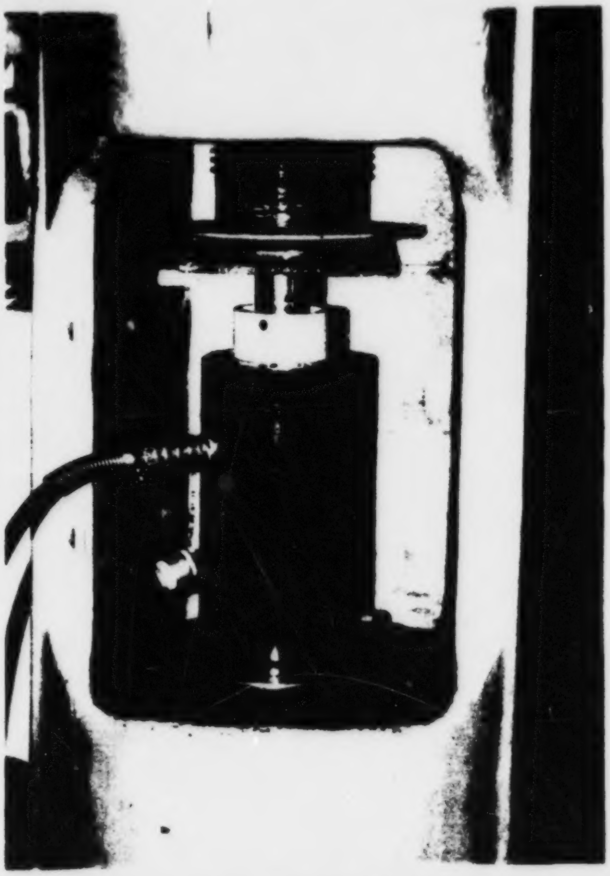
Mohr's envelopes for a range of mine fill material strengths are plotted on Figures 10 to 16, inclusive, on pages 38 to 44.

Photographs 20 and 21 on page 45 illustrate the typical failure characteristics of samples tested under cell pressures of up to 1.38 MPa.



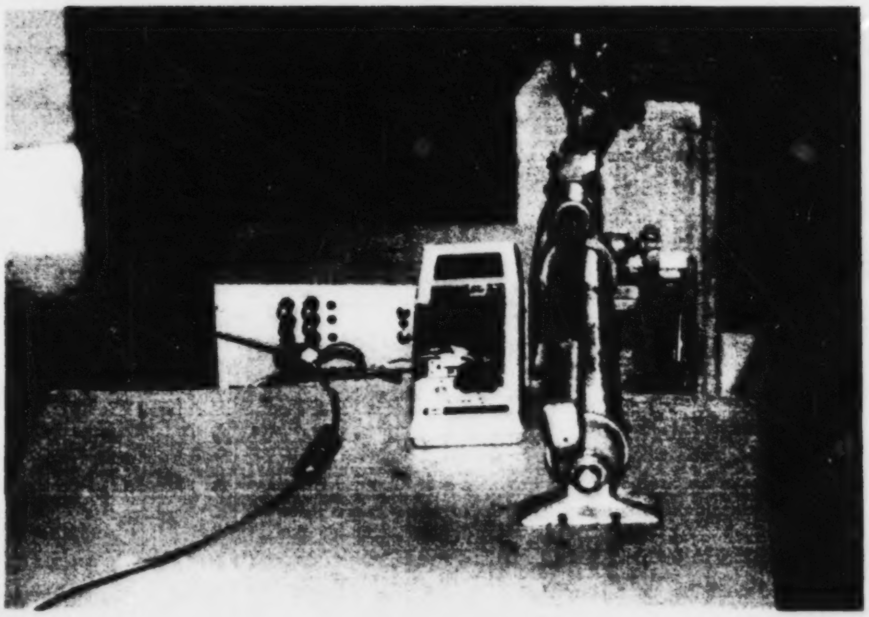
Photograph #17

Shows partially jacketed mine fill
sample with adjacent triaxial cell
equipment



Photograph #18

Sample in triaxial cell
prepared for principal
stress loading



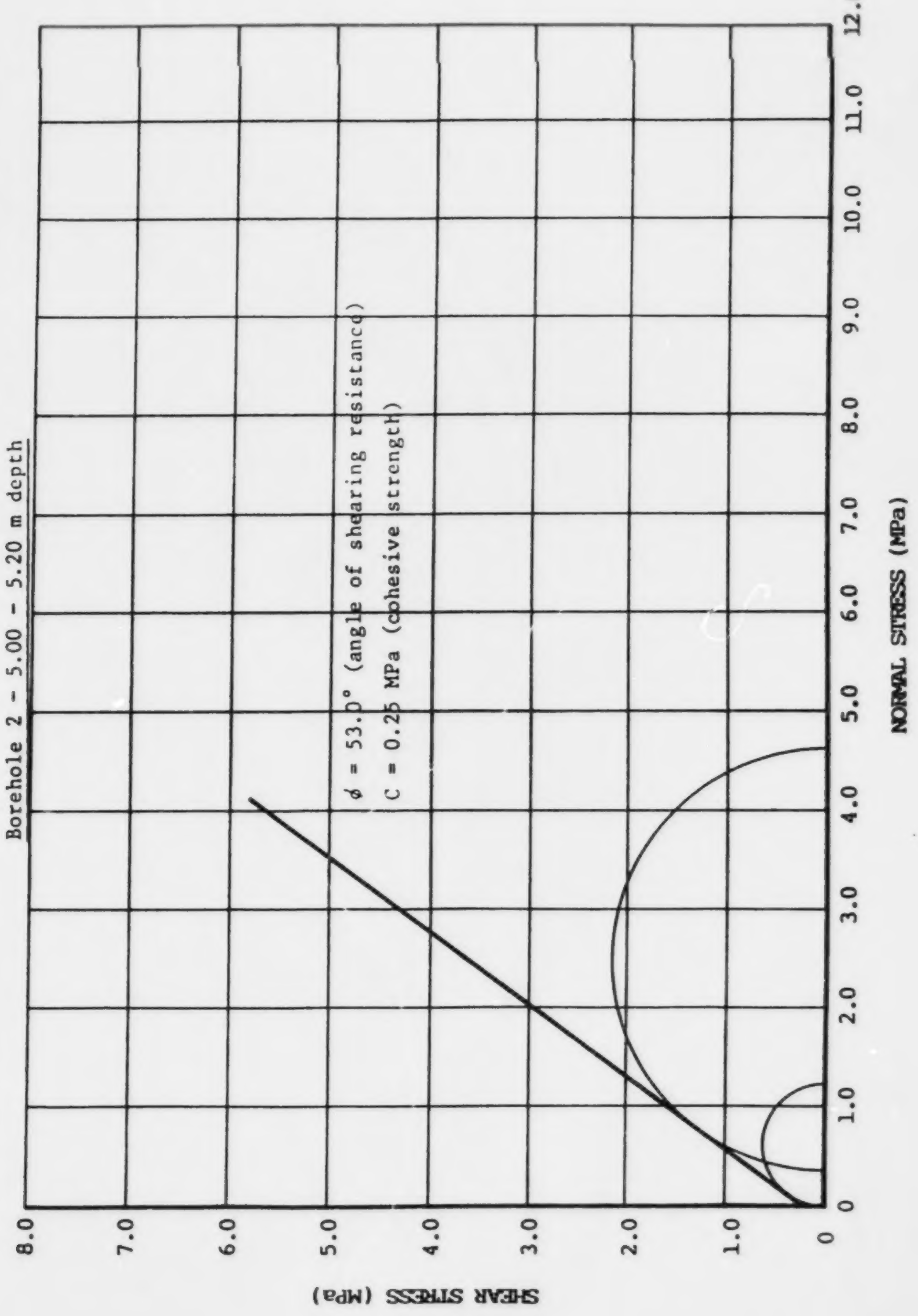
Photograph #19

Triaxial Confining Pressure Apparatus

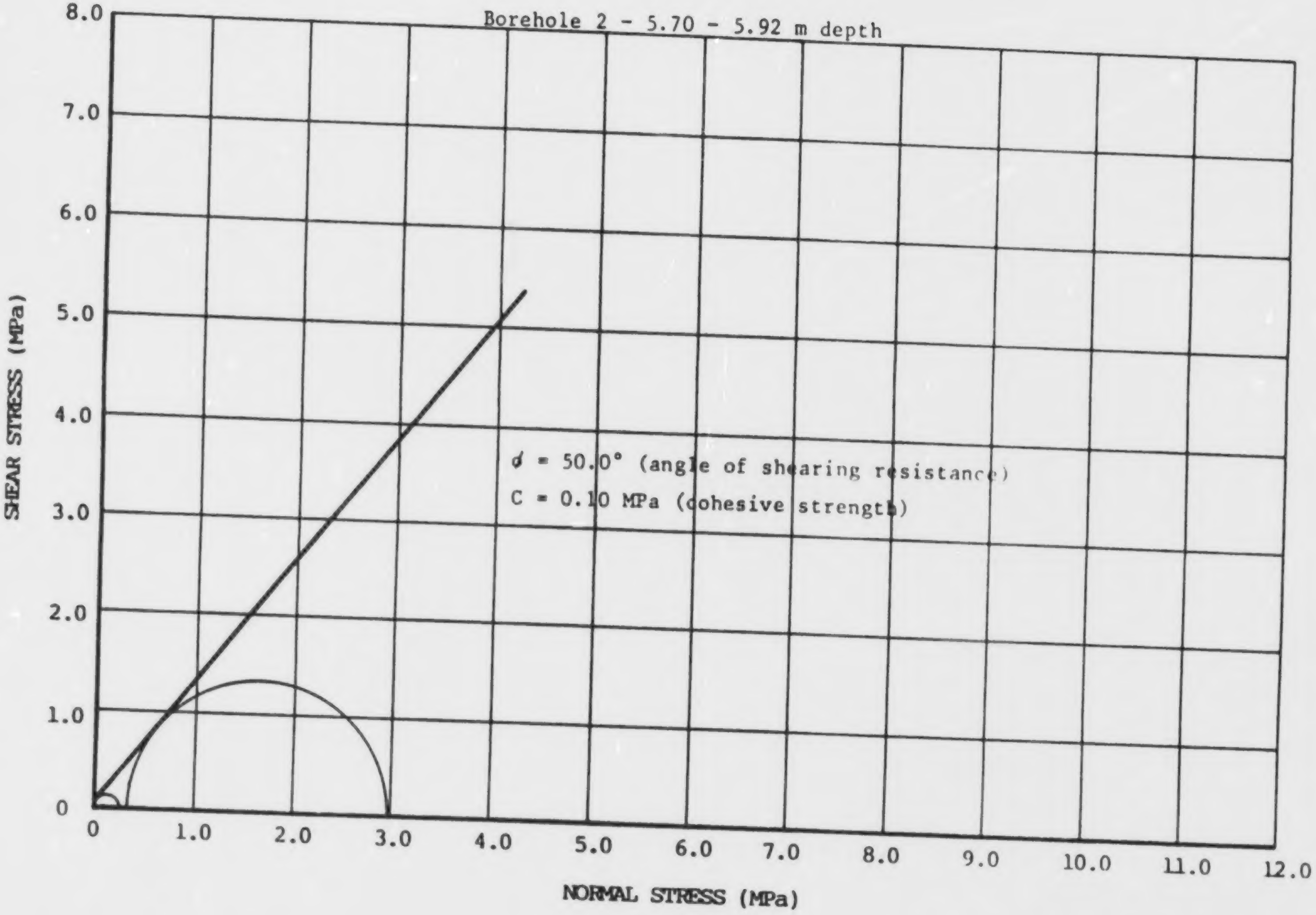
Fig. 10

## TRIAXIAL TEST

Borehole 2 - 5.00 - 5.20 m depth

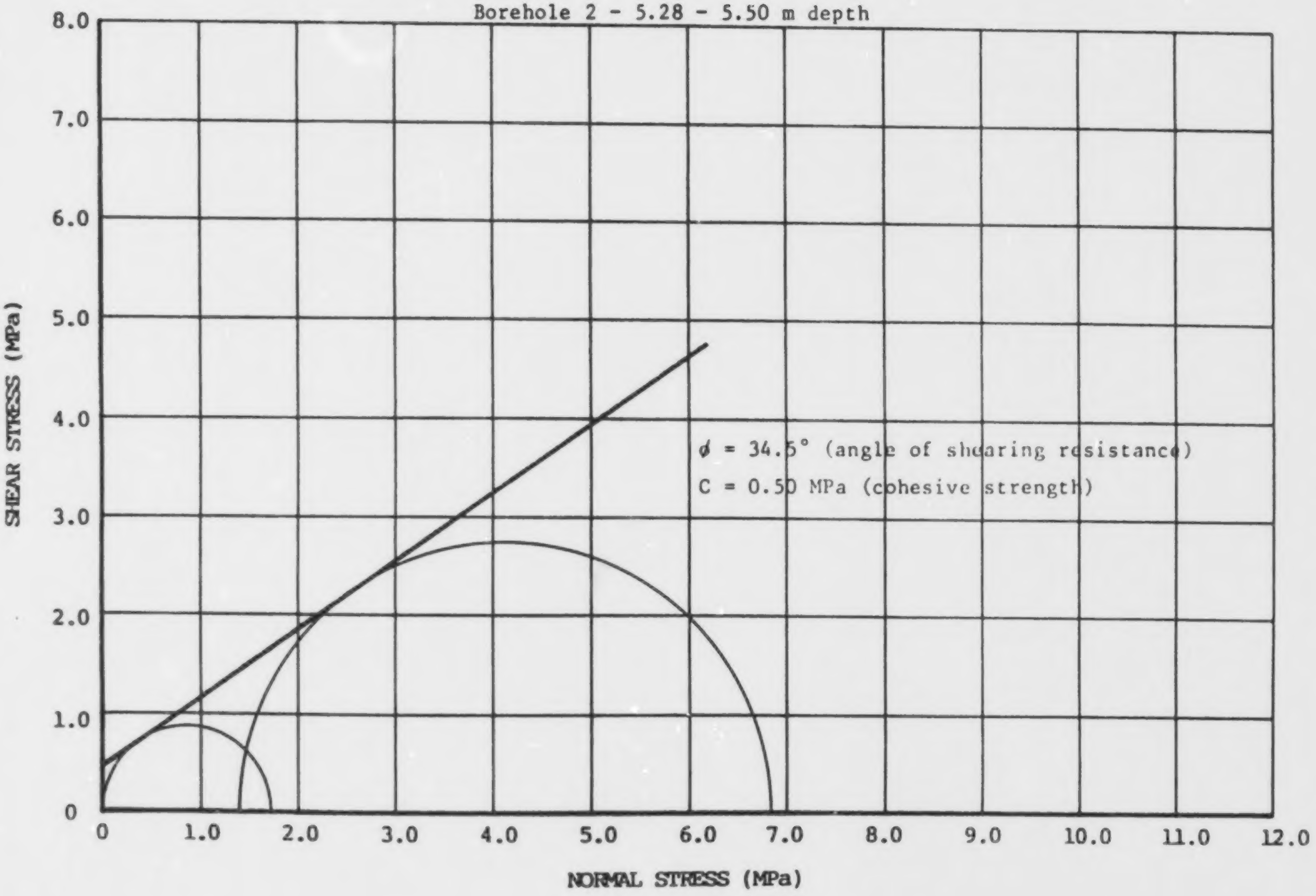


## TRIAXIAL TEST



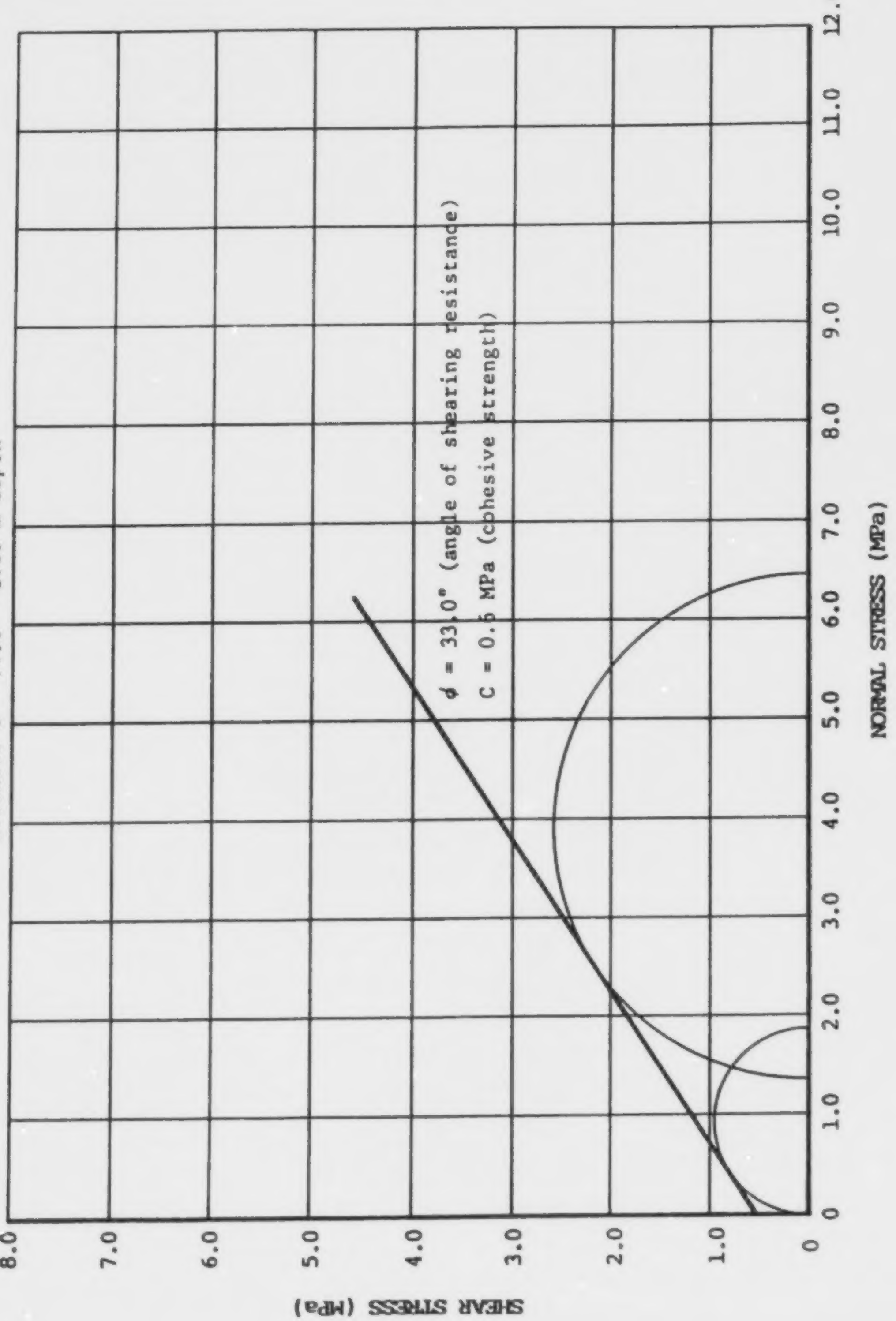
**TRIAXIAL TEST**

Borehole 2 - 5.28 - 5.50 m depth



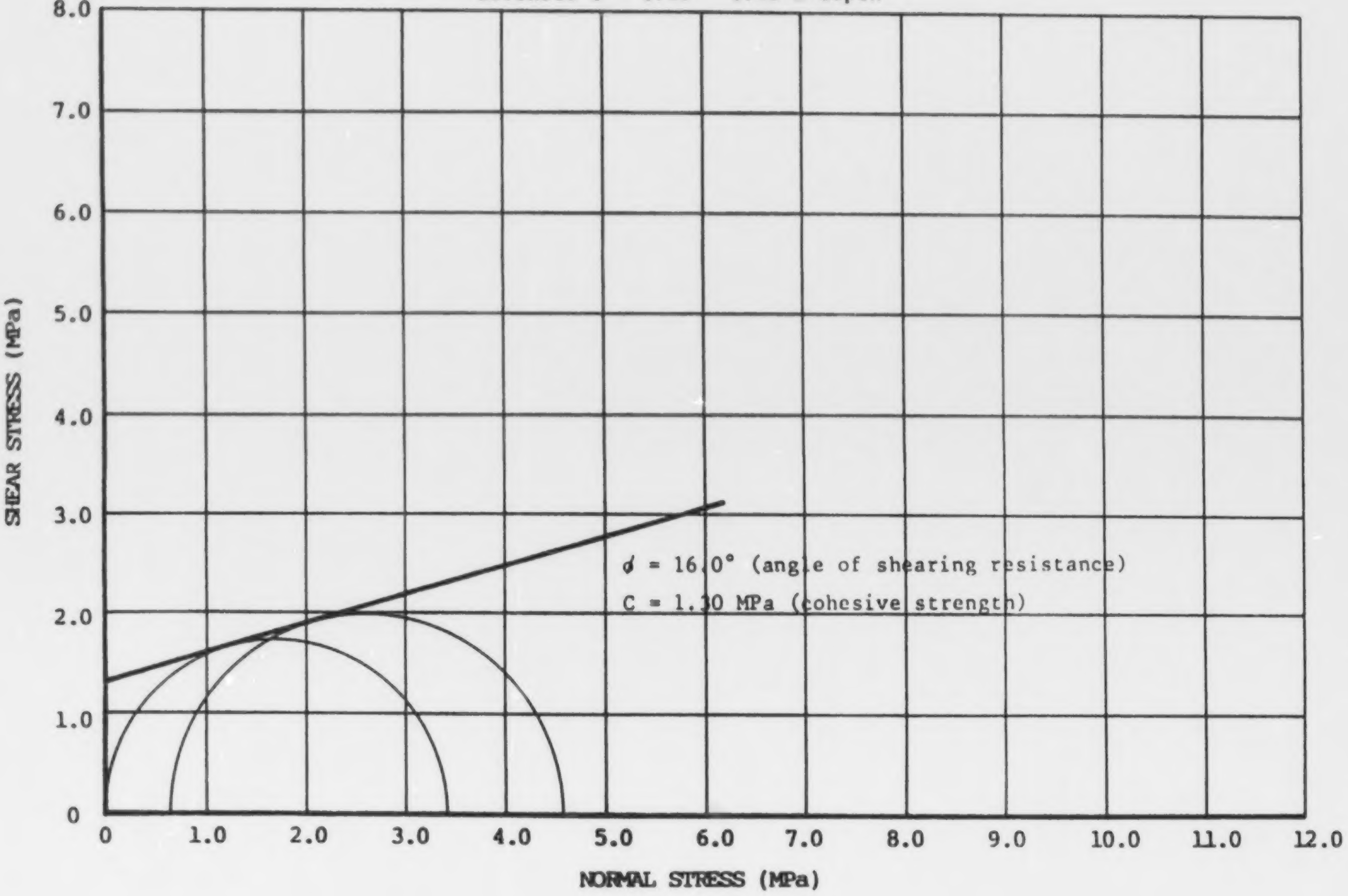
## TRIAXIAL TEST

Borehole 3 - 7.90 - 8.10 m depth



## TRIAXIAL TEST

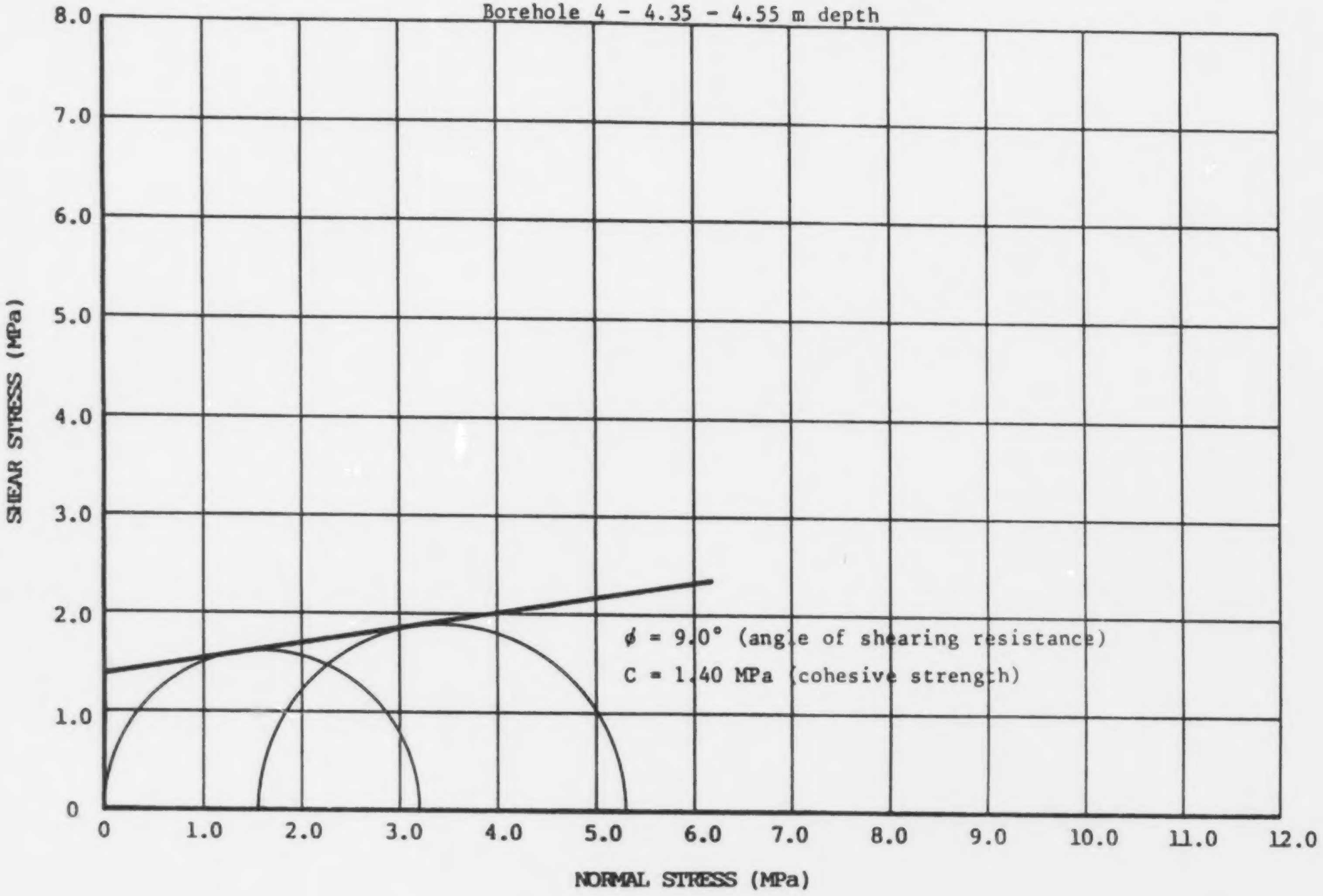
Borehole 3 - 8.16 - 8.40 m depth



ϕ = 16.0° (angle of shearing resistance)
C = 1.30 MPa (cohesive strength)

## TRIAXIAL TEST

Borehole 4 - 4.35 - 4.55 m depth



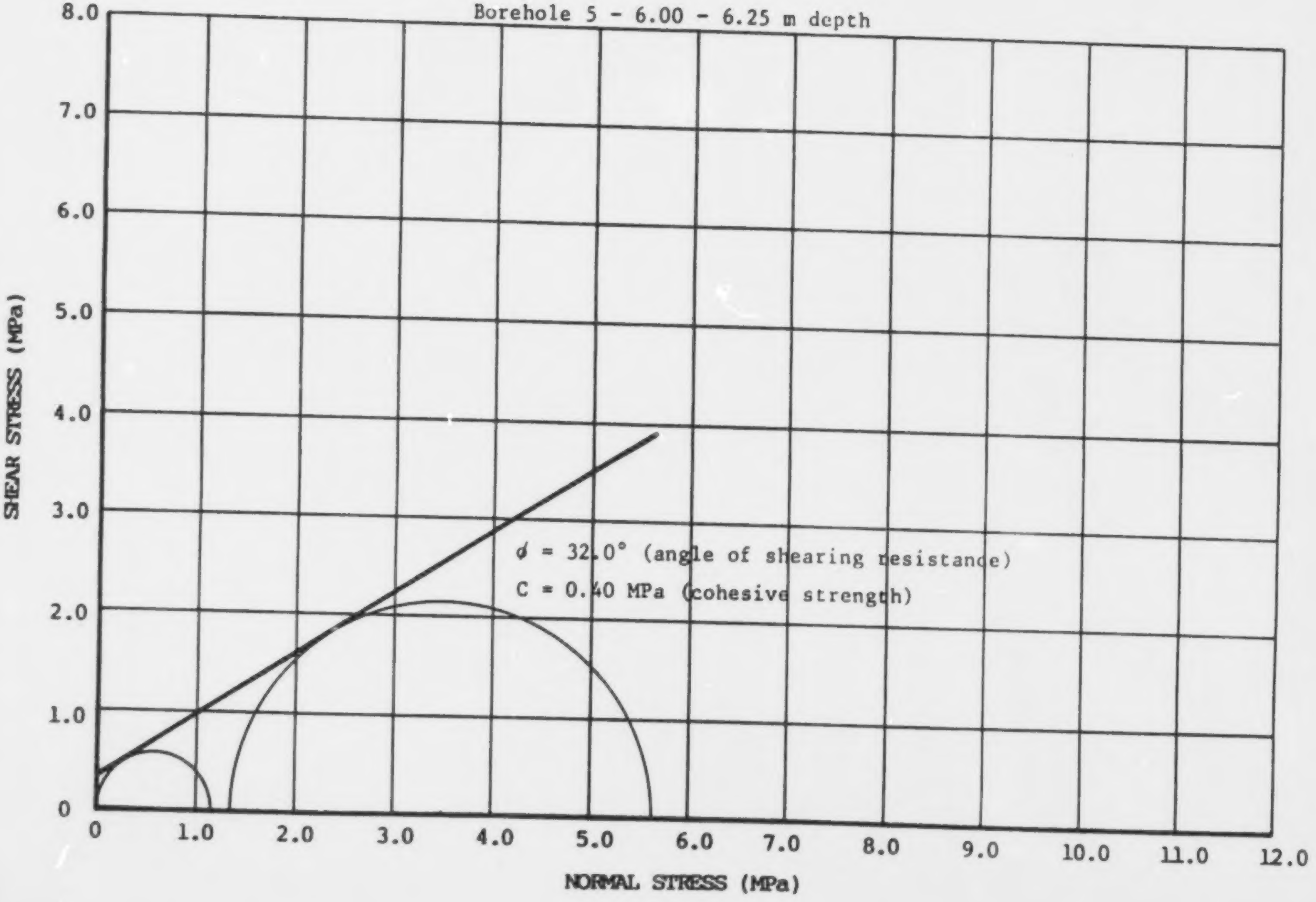
ϕ = 9.0° (angle of shearing resistance)

C = 1.40 MPa (cohesive strength)

Fig. 16

## TRIAXIAL TEST

Borehole 5 - 6.00 - 6.25 m depth



ϕ = 32.0° (angle of shearing resistance)

C = 0.40 MPa (cohesive strength)

Photograph #20

Typical failure of mine fill sample under confined triaxial loadings

Photograph #21

Typical failure of mine fill sample under confined triaxial loadings

## 7.0 DISCUSSIONS

### 7.1 Drilling and Sampling Procedures:

The drilling data summarized and illustrated in this report clearly indicates that sampling and in-situ instrumentation of cemented mine fill tailings are reasonable and obtainable objectives.

Core recovery in the order of 95 to 100%, as obtained on this project, allows for detailed visual evaluation of the deposition characteristics of the hydraulically placed mine fill (i.e., in terms of segregation, stratification, etc.). Additionally, the core can be utilized for a wide range of detailed physical and chemical testing such as cement and moisture contents, grain size analyses, unit weight determinations, uniaxial strengths and triaxial testing, etc.

Further to the sample recovery, it has also been shown that boreholes can be drilled and accessed into mine fill tailings within sufficient general dimensional tolerances for accurate in-situ instrumentation such as the pressuremeter testing.

Based on our results, however, several parameters and refinements remain to be defined with regard to mine fill drilling techniques. Together with our overall evaluation of the drilling program, the following conclusions and recommendations are summarized:

(i) Successful drilling and sampling of cemented mine fill tailings requires the use of experienced drill personnel familiar with geotechnically oriented drilling techniques. They must be conversant with the requirements for securing undisturbed samples within soft or weak overburden formations and familiar with the installation of sensitive instrumentation equipment.

(ii) Drilling requirements for weak or soft formations such as mine fill tailings require substantially differing applications of feed rates, feed pressures, torques and rpm's, as opposed to the requirements for diamond drilling of rock. Subsequently, hydraulically oriented equipment, such as the Craelius 251 unit with full variability speed controls and instrumentation of drilling pressures, is better suited to this purpose. Fixed rotational "screw-feed" type equipment is not recommended for this type of drilling application.

(iii) While core recovery on this research project was excellent, penetration of drilling fluid on the recovered samples still remains to be reduced. For the majority of the cemented tailings, the effect of the drilling fluid using a double tube sampler appears negligible. However, more weakly cemented tailings layers were obviously disturbed (i.e., occasional layers of segregated material with little or no cement content).

For this reason, the effect of the drilling fluid and sample disturbance should be evaluated on future projects.

(iv) While it was possible to maintain an acceptable borehole for general pressuremeter testing (i.e., uniform and no excessive deformations), some problems with regard to hole diameter remain to be resolved.

As noted from the pressuremeter graphs shown in Appendix 'C', the borehole diameters could have been somewhat smaller to facilitate more optimum use of the volume expansion range of the probe. Although the boreholes were advanced with an NX core barrel to accommodate the NX pressuremeter probe, the hole diameter was still slightly larger than desirable. On future projects, this problem could be mimimized by incorporating the use of more precisioned drilling apparatus.

(v) Continuous flight auger systems have shown on this project to be an effective means of securing disturbed samples for cement and moisture content determations. If properly sized, the auger method can also provide an excellent means of advancing a suitably toleranced hole for instrumentation purposes.

(vi) Application of the various drilling techniques used on this project should be equally successful in areas where mine fill must be accessed through rock. However, we would caution that parameters such as drill speeds and water pressures, etc., must be properly adjusted to accommodate for the deeper borehole depths. Additionally, it may be required that in-hole casing, etc., be used for borehole stabilization during drilling.

## 7.2 Laboratory and In-Situ Testing Data:

A summary of the laboratory and in-situ (pressuremeter) tests is included in Table 5, below.

### TABLE 5

### SUMMARY OF TEST DATA

| B.H. NO. | DEPTH (m) | M/C % AVERAGE | T/C[1] RATIO | UNIT WT. AVERAGE (kg/m³) | Q[2] (kPa) | $P_L$[3] (MPa) | $E_m$[4] (MPa) | ø[5] | C[6] (MPa) |
|---|---|---|---|---|---|---|---|---|---|
| 2 | 4.45-4.63 | 17.2 | | 2178 | --- | --- | --- | --- | --- |
| 2 | 4.69-4.88 | 17.3 | 12:1 | 2346 | --- | 13.0 | 531 | --- | --- |
| 2 | 4.94-5.12 | 28.8 | | 2130 | 1000 | --- | --- | 53.0° | 0.25 |
| 2 | 5.18-5.40 | 18.8 | | 2226 | 1600 | --- | --- | 34.5° | 0.50 |
| 2 | 5.40-5.58 | 16.3 | 15:1 | 2219 | --- | 13.0 | 235 | --- | --- |
| 2 | 5.70-5.91 | 18.0 | | 2092 | 200 | --- | --- | 50.0° | 0.10 |
| 3 | 2.59 | ---- | | ---- | ---- | 4.48 | 299 | --- | --- |
| 3 | 3.20-3.81 | 27.2 | 20:1 | ---- | ---- | 3.58 | 227 | --- | --- |
| 3 | 4.63-4.82 | 20.3 | | 2135 | 1200 | --- | --- | --- | --- |
| 3 | 4.88-5.06 | 21.2 | | 2115 | ---- | --- | --- | --- | --- |
| 3 | 5.12-5.24 | 21.8 | 13:1 | 2149 | ---- | 5.18 | 167 | --- | --- |
| 3 | 5.88-5.94 | ---- | | 2063 | 200 | --- | --- | --- | --- |
| 3 | 6.46-6.65 | 22.1 | | 2279 | ---- | ---- | --- | --- | --- |
| 3 | 6.71-6.86 | 20.7 | 9:1 | 2206 | ---- | 12.7 | 372 | --- | --- |
| 3 | 6.92-7.07 | 19.4 | | 2186 | ---- | ---- | --- | --- | --- |
| 3 | 7.93-8.11 | 22.5 | | 2142 | 1800 | ---- | --- | 33.0° | 0.60 |
| 3 | 8.17-8.29 | 24.4 | 9:1 | 2223 | ---- | 15.7 | 467 | 16.0° | 1.30 |
| 3 | 8.35-8.53 | 21.7 | | 2167 | ---- | ---- | --- | --- | --- |
| 4 | 2.20-2.35 | 26.3 | | 2236 | 600 | ---- | --- | --- | --- |
| 4 | 2.44-2.59 | 27.5 | 11:1 | 2176 | 1700 | ---- | --- | --- | --- |
| 4 | 2.65-2.80 | 29:1 | | 2059 | 600 | ---- | --- | --- | --- |
| 4 | 3.20-3.35 | 25.8 | 5:1 | 2058 | --- | ---- | 86.4 | --- | --- |
| 4 | 3.41-3.57 | 31.7 | | 2103 | 300 | ---- | --- | --- | --- |
| 4 | 3.63-3.72 | 28.3 | | 2074 | 1400 | ---- | --- | --- | --- |
| 4 | 4.12-4.20 | 22.1 | | 2157 | 2400 | 16.1 | 569 | --- | --- |
| 4 | 4.25-4.30 | 35.0 | 2:1 | 2051 | 500 | ---- | --- | --- | --- |
| 4 | 4.36-4.51 | 31.9 | | 2001 | 2700 | ---- | --- | 9.0° | 1.40 |
| 4 | 4.57-4.72 | 38.3 | | 1898 | 4200 | ---- | --- | --- | --- |
| 4 | 5.33-5.49 | 27.8 | 2:1 | 2199 | 1000 | ---- | 70.9 | --- | --- |
| 4 | 5.70 | ---- | | ---- | 300 | ---- | --- | --- | --- |
| 4 | 5.79-5.82 | 24.0 | | 2124 | ---- | ---- | --- | --- | --- |
| 4 | 5.88-6.01 | 24.1 | | 1789 | 400 | ---- | --- | --- | --- |
| 4 | 6.67-6.25 | 21.7 | 17:1 | 2160 | 100 | ---- | 77.7 | --- | --- |
| 4 | 6.30 | ---- | | ---- | 300 | ---- | --- | --- | --- |
| 4 | 6.55-6.65 | 24.3 | | 2215 | 500 | ---- | --- | --- | --- |
| 4 | 6.71-6.86 | 24.2 | 36:1 | 2210 | 100 | ---- | --- | --- | --- |

Cont'd ...

## TABLE 5 (Cont'd)

## SUMMARY OF TEST DATA

| B.H. NO. | DEPTH (m) | M/C % AVERAGE | T/C[1] RATIO | UNIT WT. AVERAGE (kg/m³) | Q[2] (kPa) | $P_\ell$[3] (MPa) | $E_m$[4] (MPa) | ∅[5] | C[6] (MPa) |
|---|---|---|---|---|---|---|---|---|---|
| 5 | 0.46-0.52 | 24.4 | 30:1 | 2073 | ---- | ---- | 44.0 | --- | --- |
| 5 | 1.22-1.28 | 27.4 | | 2103 | 200 | ---- | ---- | --- | --- |
| 5 | 1.37 | ---- | 27:1 | ---- | ---- | ---- | 12.6 | --- | --- |
| 5 | 1.52-1.62 | 26.1 | | 2137 | 30 | ---- | ---- | --- | --- |
| 5 | 1.68-2.29 | 25.8 | 36:1 | ---- | ---- | ---- | 75.4 | --- | --- |
| 5 | 2.59-3.05 | 25.0 | 31:1 | ---- | ---- | ---- | 107 | --- | --- |
| 5 | 3.51-3.57 | 28.6 | 36:1 | 1957 | ---- | ---- | 106 | --- | --- |
| 5 | 4.27-4.33 | 28.9 | | 2068 | ---- | ---- | ---- | --- | --- |
| 5 | 4.42-4.60 | 23.8 | 33:1 | 2060 | 30 | ---- | 70.4 | --- | --- |
| 5 | 4.72-4.82 | 44.2 | | 1925 | 900 | ---- | ---- | --- | --- |
| 5 | 4.88-5.01 | 26.2 | | 2275 | ---- | ---- | ---- | --- | --- |
| 5 | 5.03-5.12 | 20.0 | 11:1 | 2243 | 70 | ---- | 60.0 | --- | --- |
| 5 | 5.88-5.94 | 23.3 | 47:1 | 2106 | ---- | ---- | 68.1 | --- | --- |
| 5 | 6.00-6.16 | 21:5 | | 2002 | ---- | ---- | ---- | --- | --- |
| 5 | 6.10-6.25 | 24.6 | | 1567 | 1100 | ---- | ---- | 32.0° | 0.40 |
| 5 | 6.55 | ---- | 30:1 | ---- | ---- | 2.75 | 40.8 | --- | --- |
| 5 | 7.35-7.41 | 33.8 | | 1885 | ---- | ---- | ---- | --- | --- |
| 5 | 7.47 | ---- | 21:1 | ---- | ---- | 2.82 | 59.7 | --- | --- |
| 6 | 1.82 | ---- | | ---- | 30 | ---- | ---- | --- | --- |
| 6 | 1.83-1.92 | 28.5 | | 2067 | 10 | ---- | ---- | --- | --- |
| 6 | 1.98-2.04 | 27.9 | 16:1 | 2178 | 10 | ---- | 90.2 | --- | --- |
| 6 | 2.44-2.53 | 23.7 | | 2073 | ---- | ---- | ---- | --- | --- |
| 6 | 2.59-2.68 | 25.4 | 19:1 | 2188 | ---- | ---- | 65.8 | --- | --- |
| 6 | 3.05 | | 40:1 | ---- | ---- | ---- | 168 | --- | --- |
| 6 | 3.96-4.05 | 29.6 | 30:1 | 2146 | 100 | ---- | 129 | --- | --- |
| 6 | 4.72 | ---- | 18:1 | ---- | ---- | ---- | 94.4 | --- | --- |
| 6 | 5.64 | ---- | 18:1 | ---- | ---- | ---- | 122 | --- | --- |
| 6 | 6.40 | ---- | 11:1 | ---- | ---- | ---- | 69.6 | --- | --- |
| 6 | 7.32-7.44 | 30.3 | 19:1 | 2358 | 1000 | ---- | 77.0 | --- | --- |

*Recorded moisture contents may be affected by drill fluid.*

Notes:

[1] T/C Ratio = tailings:cement ratio
[2] Q = uniaxial strength
[3] $P_\ell$ = limit pressure (pressuremeter)
[4] $E_m$ = pressuremeter modulus
[5] ∅ = angle of shearing resistance
[6] C = cohesive strength

Correlation between the various laboratory tests is difficult, mainly because of the erratic variability in the "in-place" mine tailings backfill, with layering evident ranging from less than 10 mm up to 1000 mm thick.

Nevertheless, from a careful observation of the laboratory testing procedures, as well as from interpretation of the test data, it appears that two broad sets of physical and mechanical properties can be interpolated for the backfill. One category includes well cemented, strong fill (tailings/cement ratio of less than 10), as compared to the properties of weakly cemented fill (tailings/cement ratio greater than 25).

A tentative correlation between the test results to categorize the tailings backfill is summarized below.

(a) Strongly Cemented Tailings (T/C Ratio <10)

Uniaxial Compressive Strength (Q) - $\sim$2000 to 4000 kPa
Cohesive Strength (C) - $\sim$1000 to 1500 kPa
Angle of Shearing Resistance ($\phi$) - $\sim$10° to 15°
Average Pressuremeter Modulus ($E_m$) - $\sim$400 to 600 MPa
Average Pressuremeter Limit Pressure ($P_\ell$) - $\sim$10 to 20 MPa
Estimated $E_m/P_\ell$ ratio - $\sim$30 to 40

Although not specifically tabulated above, an estimate of Young's Modulus was calculated from the elastic portion of the stress/strain curves during uniaxial and triaxial testing. The strain at failure (generally a brittle, shear failure) for the strongly cemented tailings was normally in the order of 1% or less. Young's Modulus was found to lie within the range:

$$E = 300 \text{ MPa to } 500 \text{ MPa}$$

These values can be compared, for example, with a Young's Modulus of 100 MPa to 200 MPa for very dense sand and gravel, and 14,000 MPa for limestone bedrock.

(b) Weakly Cemented Tailings (T/C Ratio >25)

Uniaxial Compressive Strength (Q) - ∿200 to 1000 kPa
Cohesive Strength (C) - ∿100 to 400 kPa
Angle of Shearing Resistance ($\phi$) - ∿35° to 50°
Average Pressuremeter Modulus ($E_m$) - ∿40 to 200 MPa
Average Pressuremeter Limit Pressure ($P_\ell$) - ∿5 to 15 MPa
Estimated $E_m/P_\ell$ Ratio - ∿8 to 15

The Young's Modulus estimated from the elastic portion of the stress/strain curves obtained during uniaxial and triaxial testing was found to be within the range:

$$E = 50 \text{ MPa to } 100 \text{ MPa}$$

These values can be compared with typical Young's Modulus of 10 MPa to 50 MPa for loose to dense sands.

An evaluation of the laboratory tests, as performed on samples extracted from "in-place" tailings, was found to be in general agreement with corresponding parameters derived from previous testing on tailings prepared within uniform, controlled, laboratory conditions*.

It is noted that the uniaxial strengths measured for strongly cemented tailings fall into the 2000 kPa to 4000 kPa range (T/C ratio <10), whereas for the weaker fill, uniaxial strengths in the order of 200 kPa to 400 kPa range (T/C ratio >20) were obtained. These parameters are similar to the empirical strengths used by Falconbridge and illustrated previously in this report in Figures 4 and 5 on pages 5 and 6, respectively.

---

**Laboratory Studies of Cement - Stabilized Mine Tailings*, W.S. Weaver and R. Luka, CIM Transactions: Vol. LXXIII, pp 204 - 217, 1970*

In the triaxial tests, no correction of the axial stress for pore pressure or provisions for drainage of specimens were made. The lateral (cell) stresses applied to the samples in the triaxial tests were limited to a maximum of about 4000 kPa. Much higher cell stresses were attempted, i.e., up to 10,000 kPa; however, under these higher confining pressures, almost immediate consolidation of the triaxial samples was observed with a marked reduction in sample volume and a corresponding "bleeding" of pore water. Subsequent interpretation of failure loads for the "consolidated" samples proved to be difficult. Consequently, the feasibility of performing triaxial tests with cell pressure exceeding about 5000 kPa is questionable, unless the pore water pressure can be measured during testing.

We should caution that evaluation and correlation of the various laboratory test results is difficult and the variability in results can often be attributed to the erratic changes in mine tailings strengths, gradations and moisture contents, etc., within the stratifications evident in the "in-place" fill.

The pressuremeter tests also indicate a similar trend, comparing the values of modulus and limit pressures in the strongly cemented zones with the weaker layers. Typical values of $E_m$ and $P_\ell$ for soil and rock materials, based on data supplied in the Canadian Foundation Engineering Manual, are illustrated below in Table 6.

## TABLE 6

## TYPICAL PRESSUREMETER VALUES

## (Canadian Foundation Engineering Manual - Part 1)

| TYPE OF SOIL/ROCK | $E_m$ (MPa) | $P_\ell$ (MPa) | $E_m/P_\ell$ |
|---|---|---|---|
| loose silty sands | 0.5 - 2.0 | 0.1 to 0.5 | $\sim$5 |
| weakly cemented glacial till | 7.5 to 40.0 | 1.0 to 5.0 | $\sim$8 |
| limestone | 80 to 10,000 | 3 to 100 | $\sim$50 |

The results of the in-situ pressuremeter tests indicate that the mine fill, with a tailings/cement ratio of less than 10, is stronger than a dense glacial till and approaches the characteristics of a weak limestone. On the other hand, the mine fill, with tailings/cement ratios of greater than 20, can be compared with a moderately cemented glacial till and/or a very dense, well graded, sand and gravel. Similar values for pressuremeter parameters have also been obtained in generally weaker mine tailings fills by others*.

## 8.0 SUMMARY

During this project, it has been demonstrated that, with careful control and techniques, boreholes can be accurately drilled into cement/tailings mine backfill. Moreover, core recovery was excellent, with a percentage of recovery in the 95% to 100% range, even in weak layers. The recovered cores could be examined, classified and used for a variety of laboratory testing, i.e., physical and chemical tests including uniaxial and triaxial tests.

The pressuremeter has also proved to be an effective method in evaluating the in-situ parameters and characteristics of the mine fill.

Interpretation of the data is difficult because of the erratic variability of the in-situ fill. In addition, the amount of laboratory testing and instrumentation for this particular project was limited to the scope of work required in the original terms of reference, as well as budget constraints and scheduling during the underground drilling and instrumentation phase. Undoubtedly, further refinement and improvement in procedures can be made, both in the drilling techniques, as well as laboratory testing, particularly triaxial testing. These improvements and modifications should be investigated further.

---

**Development of an In-Situ Testing Technique for Backfill in Underground Mines, M. Scobie, L. Piciacchia, J.F. Capelli, Proceedings of the 39th Geotechnical Conference, Ottawa, 1986*

The results obtained to date are nevertheless encouraging. Ultimately, it may be possible to evaluate the effectiveness of backfill support characteristics in mines using semi-empirical and rapid in-situ methods such as Proctor penetrometers, the Windsor probe, or pull out testing techniques. The success of any such methods is, however, questionable because of the erratic variations within the in-situ backfill.

In addition, before any such procedures can be accepted, further research and development will be necessary. This additional laboratory and field data will enhance confidence in the evaluation of in-situ fills with the purpose of eventual acceptability of rapid semi-empirical methods for evaluating cement/tailings mines backfill. Placement techniques should also be studied to improve the uniformity of the backfill and to facilitate field, laboratory and analytical evaluation.

Yours very truly,
TROW ONTARIO LTD.

M. H. Dufresne
for: Dennis A. Pigeau
Materials Manager

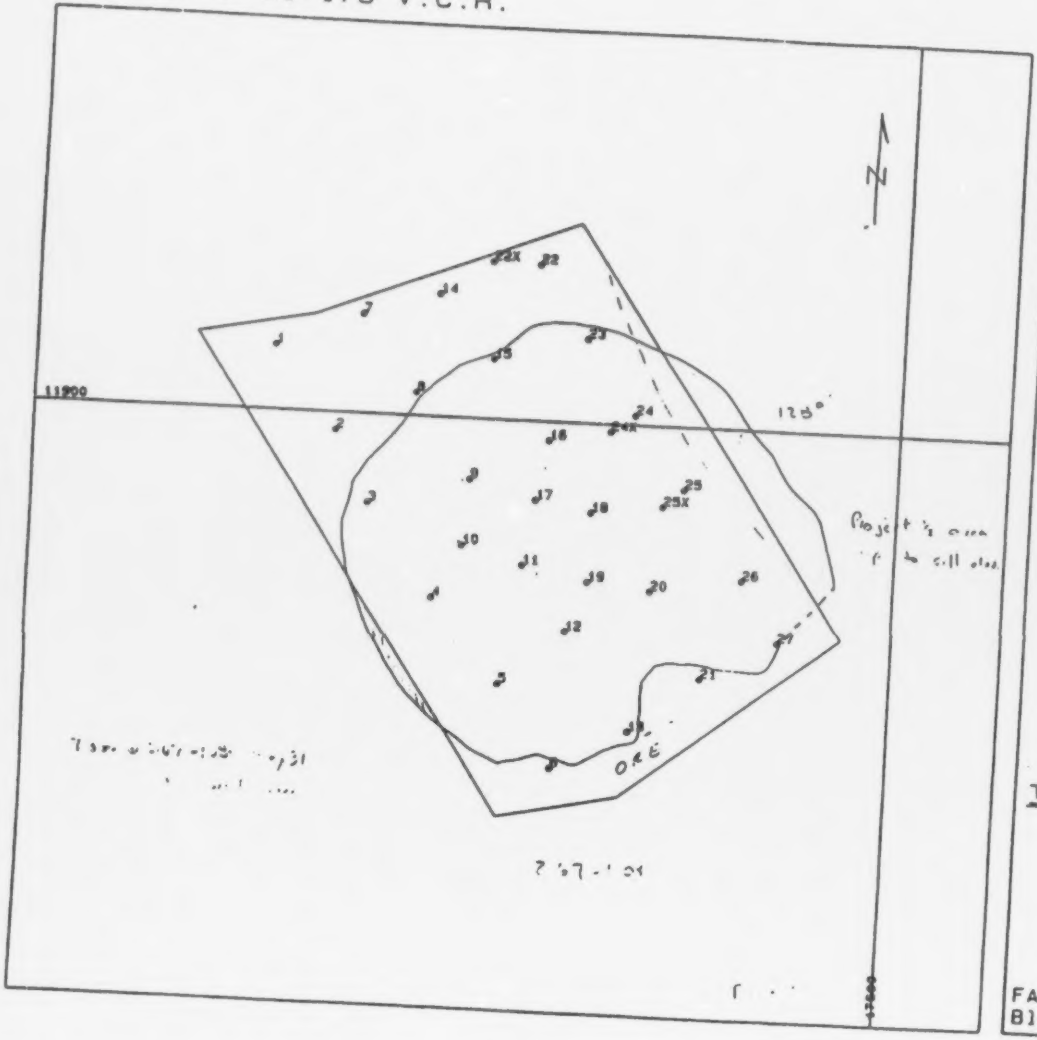
I. W. Gore, P. Eng.

DAP:gmv
Encl.
Dist: Falconbridge Limited (3)

# APPENDIX D

## STOPE VOID PROFILES

Title : 31-176 V.C.R.


N
11900
12800
128°
1
2
3
4
5
7
8
9
10
11
12
13
14
15
16
17
18
19
20
21
22
22X
23
24
24X
25
25X
26
27
ORE
Project ½ area
up to sill elev.
2.67 - 1.04

Scale : 10
Plan Elevation : 10054.22

**Rotation Angles**

Z axis : -35
Y axis : 0
X axis : -90

Ore = $\frac{1155 m^2 \times 44}{8.26}$ = 6153 T.

2.67 - 1.08

Total:

615. 2.67 1.08
886 0.45 - 1.12 [illegible]
15019 1.60 - 1.10
1051 - -
307 0.10 0.11
16377 1.47 - 1.01

FALCONBRIDGE LTD.
Blasthole Design Package

## Notes

Pencil Outline Elev. 10055.5

Outline picked up with V.C.R. Void Measurement Device

Dilution

T.S.F. [illegible]

T.S.F. [illegible]

Total

T.S.F.

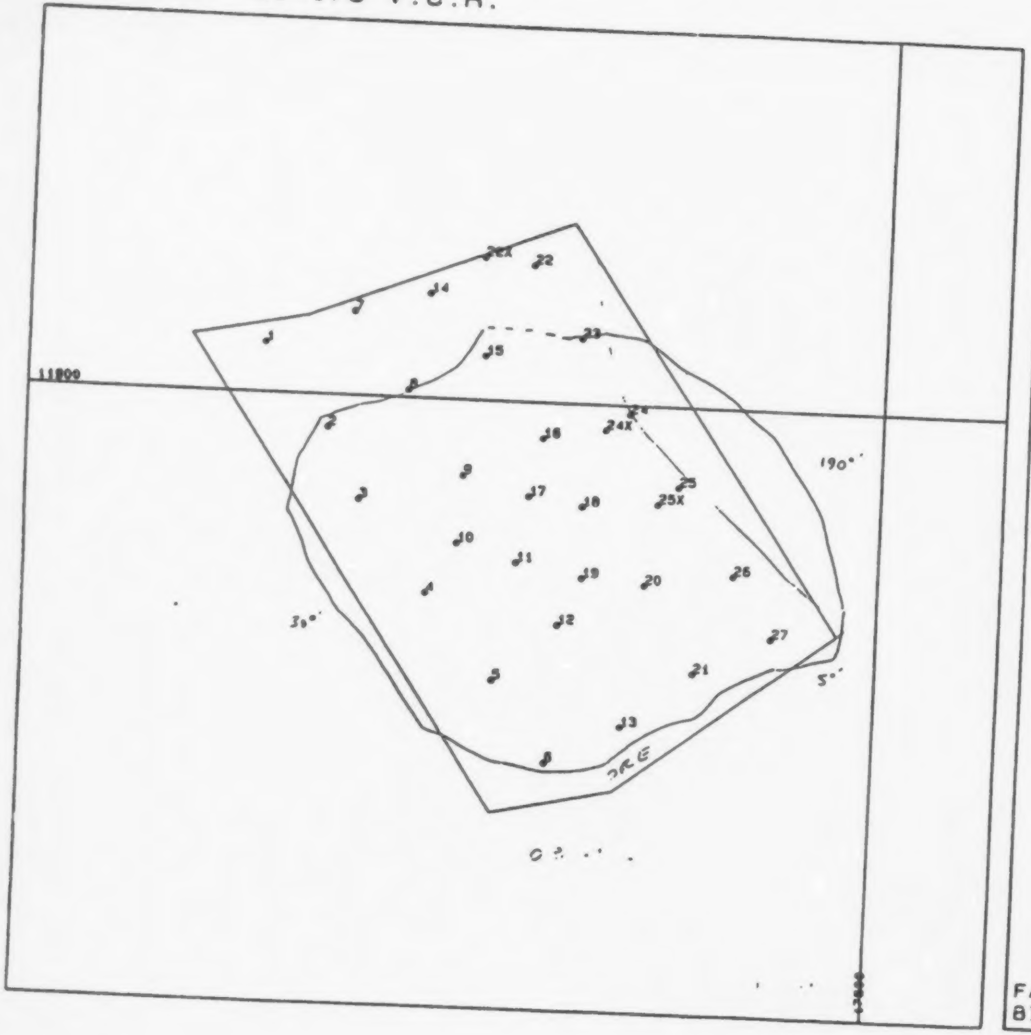
Waste

1051 Tons

[illegible]

Dil. [illegible]

Title : 31-176 V.C.R.


11800
22X
22
14
15
23
24
24X
16
17
18
25
25X
10
11
19
20
26
12
27
21
13
190°
30°
5°
ORE

Scale : 10
Plan Elevation : 10045.05

Rotation Angles
Z axis : -35
Y axis : 0
X axis : -90

Ore
1190° / [illegible] = 1207 T
9.82

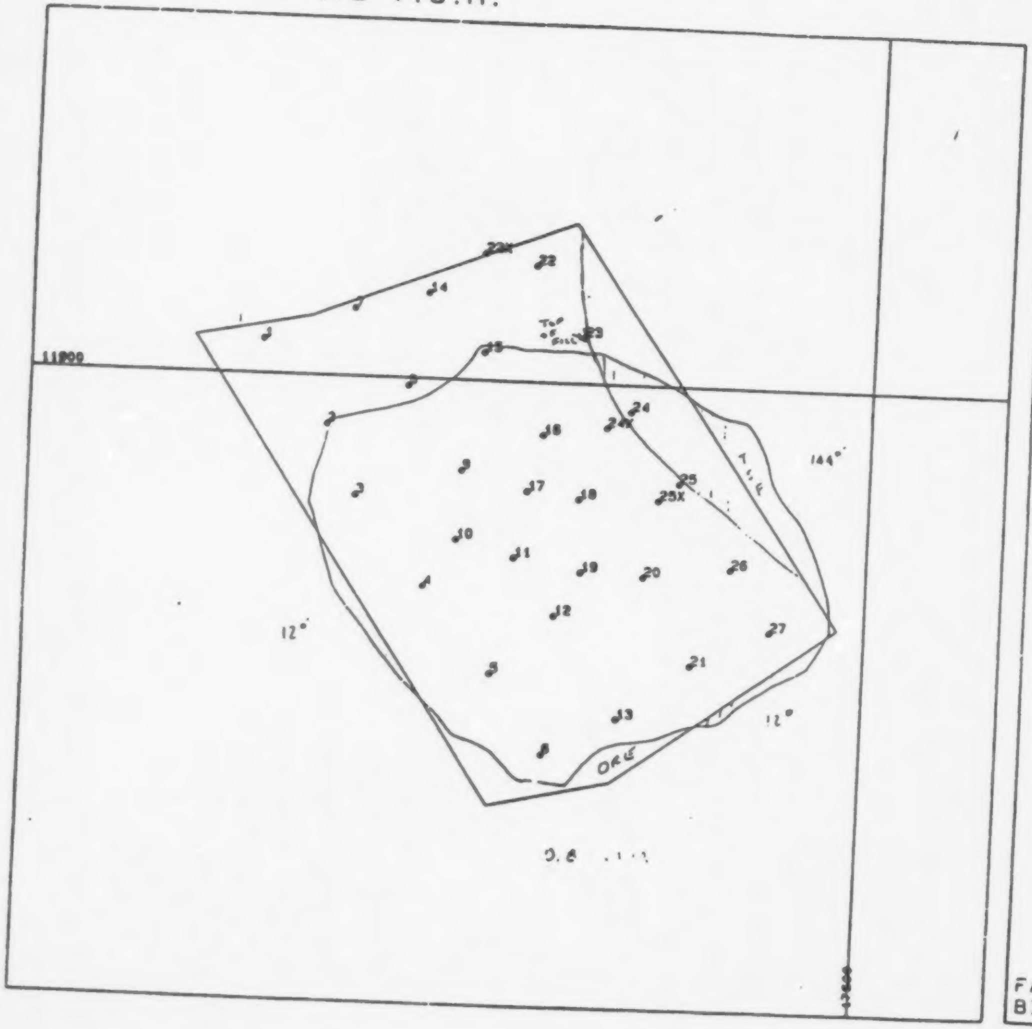
FALCONBRIDGE LTD.
Blasthole Design Package

Notes

Pencil Outline Elev: 10045:
Outline picked up with user
Void Measurement Device

Dilution
T.S.F. 270 × 2 / 20 = [illegible] T.

Title : 31-176 V.C.R.



11800
22X
22
14
TSF
23
15
24
24X
16
9
17
18
25
25X
T.S.F
144°
10
11
19
20
26
12°
12
27
21
12°
13
ORE
0.6

Scale : 10
Plan Elevation : 10035.88

Rotation Angles
Z axis : -35
Y axis : 0
X axis : -90

Ore

1166 × [illegible] : 1160 T.
98%

FALCONBRIDGE LTD.
Blasthole Design Package

## Notes

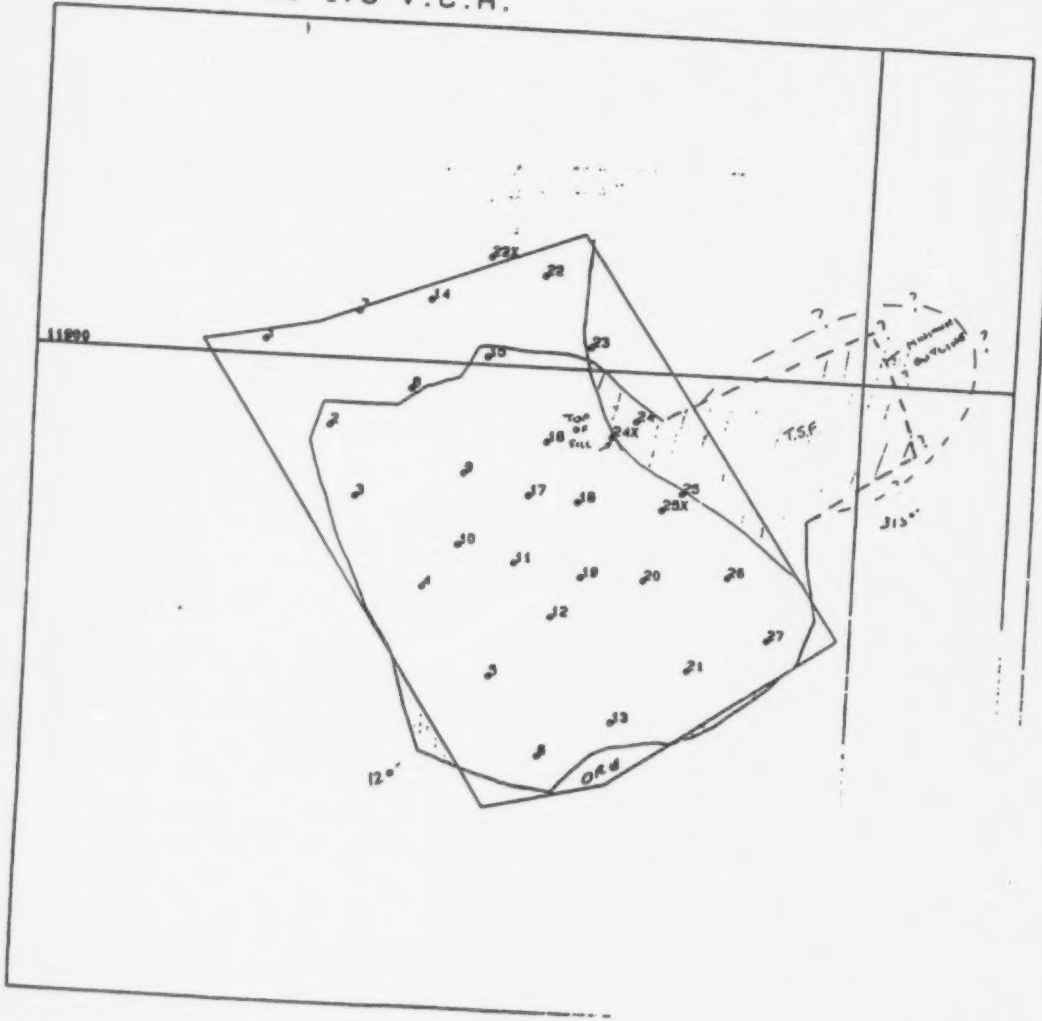
[illegible]

Dilution

T.S.F. [illegible]

Waste [illegible]
10.71

Title : 31-176 V.C.R.


11800
22X
22
14
23
15
Top of fill
16
24
24X
T.S.F
Maximum outline
?
25
25X
17
18
10
11
19
20
26
12
27
21
13
Ore
12°
315°

Scale : 10
Plan Elevation : 10024.42

**Rotation Angles**
Z axis : -35
Y axis : 0
X axis : -90

Notes

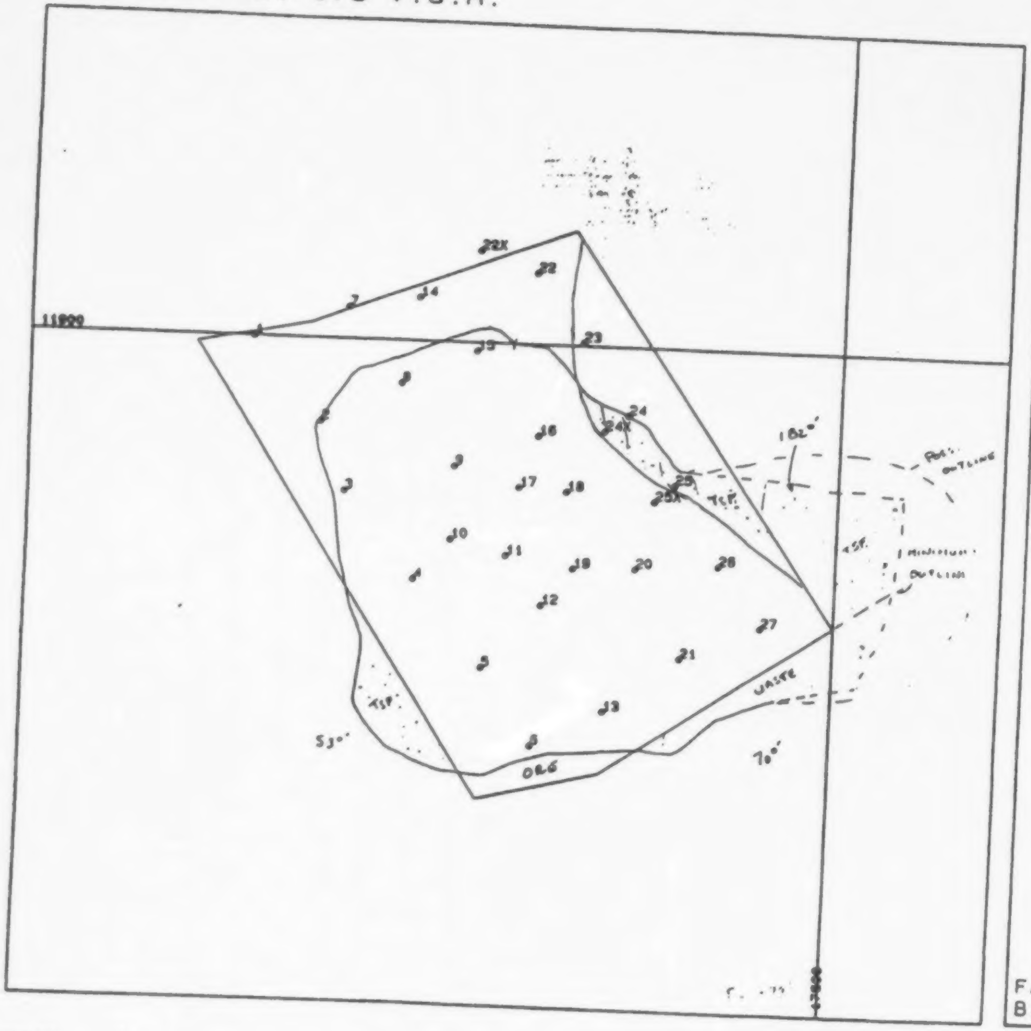
[illegible] Outline Elev. 100[illegible]
Outline picked up by
[illegible] [illegible] measurement

Ore

[illegible]

[illegible] = 163 T

Title : 31-176 V.C.R.


11800
22X
22
14
7
23
15
24
24X
16
17
18
25
25X
T.S.F.
182°
POSS. OUTLINE
10
11
19
20
28
T.S.F.
MINIMUM OUTLINE
12
27
21
WASTE
T.S.F.
13
53°
70°
ORE

Scale : 10
Plan Elevation : 10015.25

Rotation Angles
Z axis : -35
Y axis : 0
X axis : -90

Ore

$\frac{1161^{\circ} \times 10}{9.86}$ = 1157 T.

FALCONBRIDGE LTD.
Blasthole Design Package

Notes

Pencil Outline Elev: 10015

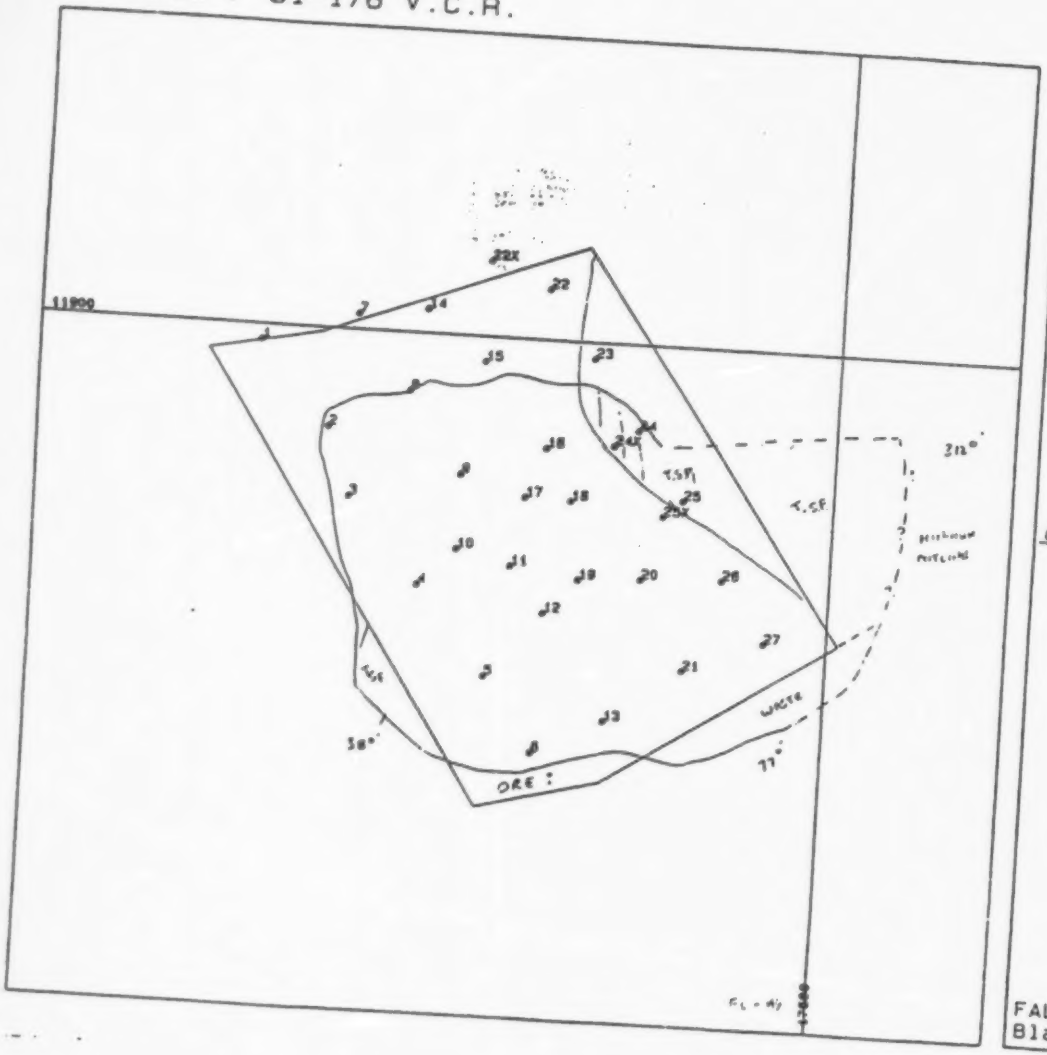
Outline picked up with U.S.B.M
Void Measurement Device

Dilution

T.S.F. $\frac{235^{\circ} \times 10}{20}$ = 118 T.

Waste $\frac{70 \times 10}{10.72}$ = 65 T

Title : 31-176 V.C.R.


11900
22X
22
7
24
1
15
23
8
2
18
24X
24
3
17
T.S.F
25
25X
312°
T.S.F
10
11
19
20
26
4
12
27
T.S.F
5
21
13
WASTE
38°
6
77°
ORE ?

Scale : 10
Plan Elevation : 10006.08

**Rotation Angles**

Z axis : -35
Y axis : 0
X axis : -90

Ore:

$$\frac{1085^{\square'} \times 10}{9.86} = 1100 \text{ T.}$$

FALCONBRIDGE LTD.
Blasthole Design Package

## Notes

Pencil Outline Elev: 10005.5

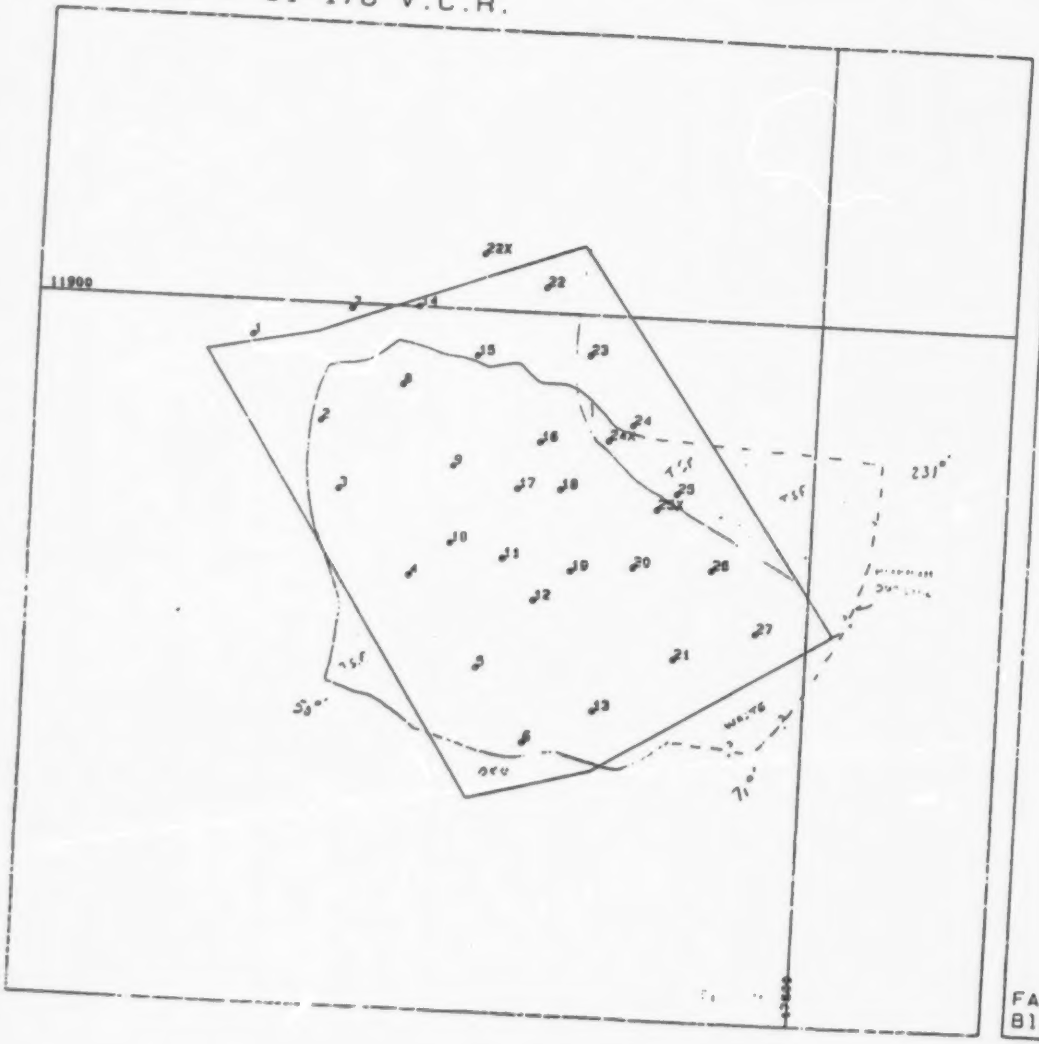
Outline picked up with U.S.3.11 Void Measurement Work

Dilution

T.S.F $\frac{360^{\square'} \times 10'}{20}$ = 180 T.

Waste $\frac{77 \times 10}{10.72}$ = 72 T.

Title : 31-176 V.C.R.


11900
22X
22
23
24
24X
25
25X
26
27
21
20
19
18
17
16
15
14
13
12
11
10
9
8
7
6
5
4
3
2
1
231°
50°
71°
WASTE

Scale : 10
Plan Elevation : 9994.624

Rotation Angles
Z axis : -35
Y axis : 0
X axis : -90

Ore

1132 [illegible]
— 9 3 [illegible]

FALCONBRIDGE LTD.
Blasthole Design Package

Notes

Pencil Outline Glos. 9995.5

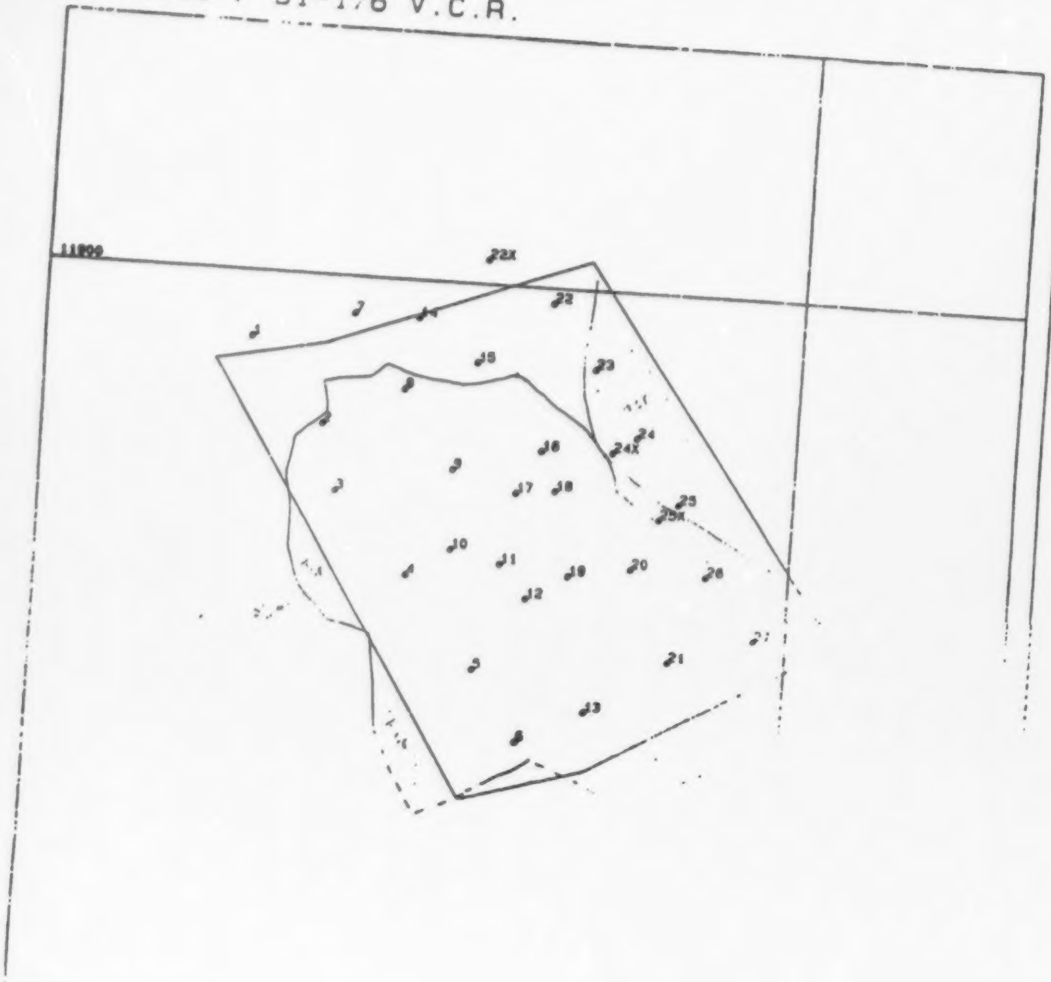
[illegible]

Dilution

T [illegible]

Waste [illegible]

Title : 31-176 V.C.R.



11800
22X
22
23
24
24X
25
25X
15
16
17
18
19
20
21
26
27
10
11
12
13

Scale : 10
Plan Elevation : 9976.288

**Rotation Angles**
Z axis : -35
Y axis : 0
X axis : -90

Ore

[illegible]

## Notes

[illegible]

D.l.t.:

[illegible] 24T.

[illegible]

# APPENDIX E

## DRILL CORE MAPPING

| | | | |
|---|---|---|---|
| FALCONBRIDGE MINE BACKFILL<br>CANMET SILL MAT PROJECT | DIP -62°<br>AZIMUTH 259° | LOGGED BY: A. BRADFIELD<br>SEPT. 23, 1986 | BOREHOLE<br>E1 |
| LOCKERBY MINE<br>2950 LEVEL | DRILLED BY CRAELIUS<br>DIAMEC 251/260 | | SHEET<br>1 OF 1 |

| SAMPLE NO. | FOOTAGE (FT.) | DESCRIPTION | REC. (%) | COMP. STR. (psi) | TAILS: CEMENT | MOI (% |
|---|---|---|---|---|---|---|
| | ROCK | | | | | |
| 243 | 50 | Dark black, sand size, extremely weak, very moist, no banding, no cement, all loose. | 50 | | 24:1 | 19 |
| Anchor | 55 | Grey, silt size, med. strength & moisture, brittle, one 4" section, rest loose. | 10 | | | 21 |
| 244 | 60<br>65 | Grey, silt size, med. strength & moisture, brittle to 62'. 62' to 63' light grey, clay size, high cement & strength. 64' to 65' dark grey, sand size, high moisture, low strength, loose. | 10<br>80 | 75.4 | 47:1 | 11 |
| | 70 | Grey, silt size, med. strength & moisture. 3" section of waste rock in box, all loose. Med. cement. | 10 | | 8.9:1 | 27. |
| 245 | 75 | Grey, silt size, slightly higher than med. strength med. moisture, 1" section of waste rock, 5 x 1"-3" sections, rest loose. Med. cement. | 20 | | | |
| Anchor | 80 | Slightly darker than grey, sand size, low strength, med. moisture, 2" & 4" & 6" sections, low cement. | 20 | 32.7 | | 35. |
| Anchor | 85<br>END | Same as previous | 15 | | | 22.<br>22. |
| | | OVERALL RECOVERY | 26 | | | |

REMARKS: PERSONAL CONVENTIONS USED

| | | | |
|---|---|---|---|
| AVG. RANGES FOR COMP. STR. | LOW 0 - 50 psi | SIZES: | Low tails, high cement = clay size |
| | MED. 50 - 100 psi | | Med tails, med. cement = silt size |
| | HIGH > 100 psi | | High tails, low cement = sand size |

86-248

| | | | |
|---|---|---|---|
| FALCONBRIDGE MINE BACKFILL<br>CANMET SILL MAT PROJECT | DIP -68°<br>AZIMUTH 250.5° | LOGGED BY: A. BRADFIELD<br>SEPT. 23, 1986 | BOREHOLE<br>E3 |
| LOCKERBY MINE<br>2950 LEVEL | DRILLED BY CRAELIUS<br>DIAMEC 251/260 | | SHEET<br>1 OF 1 |

| SAMPLE NO. | FOOTAGE (FT.) | DESCRIPTION | REC. (%) | COMP. STR. (psi) | TAILS: CEMENT | MO ( |
|---|---|---|---|---|---|---|
| | 75 | | | | | |
| | ROCK | | | | | |
| Anchor | 80 | No sample available | | | | |
| | | Sl. darker than grey, sand size, med. str. & moist. | 95 | | | |
| | 85 | | | | | |
| | | Grey, silt size, high str., dry | | | | |
| | | Lt. grey, clay size, very high str., very dry | 80 | | | |
| Anchor | 90 | Drk. than grey, sand size, med. str. & moist. | | | | |
| 246 | | Lt. grey, silt size, very high str. & cem, very dry | | | | 2 |
| 247 | | Gr. silt size, high str. & cem., very dry | | | 11:1 | 1 |
| Anchor | 95 | Drk. than grey, sand size, low str. & moist. | 80 | | 54:1 | 1 |
| | | Grey, silt size, high str., med. cem., low moist. | | | | |
| | END | Black, sand, very moist, very low str., no cem. | | | | |
| | 100 | | | | | |
| | | OVERALL RECOVERY | 86 | | | |

REMARKS: PERSONAL CONVENTIONS USED

| AVG.RANGES FOR COMP.STR. | | SIZES: |
|---|---|---|
| LOW | 0 - 50 psi | Low tails, high cement = clay size |
| MED. | 50 - 100 psi | Med tails, med. cement = silt size |
| HIGH | > 100 psi | High tails, low cement = sand size |

| | | | |
|---|---|---|---|
| FALCONBRIDGE MINE BACKFILL<br>CANMET SILL MAT PROJECT | DIP 0°<br>AZIMUTH 325° | LOGGED BY: A. BRADFIELD<br>SEPT. 23, 1986 | BOREHOLE<br>G1 |
| LOCKERBY MINE<br>3125 LEVEL | DRILLED BY CRAELIUS<br>DIAMEC 251/260 | | SHEET<br>1 OF 1 |

| SAMPLE NO. | FOOTAGE (FT.) | DESCRIPTION | REC. (%) | COMP. STR. (psi) | TAILS: CEMENT | MOI ( |
|---|---|---|---|---|---|---|
| | | Grey, silt size, high str.&cem., med. moist. 20% loose | | | | |
| | | Lt.gr.,cl.size,med.str.,high cem.,low moist. all loose | 60 | | | |
| | 5 | | | | | |
| | | Gr., silt size, med. str. & cem. & moist., all loose | 30 | | | |
| | 10 | Lt. gr., cl. size, med. str., high cem., low moist. | | | | |
| | | Gr., silt size, med. str. & cem. & moist., 50% loose, | 50 | | | |
| | 15 | only 15% of last 2' recovered | | | | |
| 255 | | Grey, clay to silt size, high strength & cement, med. | 80 | 130.3 | 14:1 | 17 |
| | 20 | moisture, 30% loose | | | | |
| | | Grey, clay to silt size, high strength & cement, med. | | | | |
| 256 | | moisture, 50% loose | 70 | 137.5 | 8.1:1 | 14 |
| | 25 | Grey, clay to silt size, high strength & cement, med. | | | | |
| | | moisture, all loose | 50 | | | |
| | END30 | | | | | |
| | | OVERALL RECOVERY | 57 | | | |

REMARKS: PERSONAL CONVENTIONS USED

| AVG.RANGES FOR COMP.STR. | | SIZES: |
|---|---|---|
| LOW | 0 - 50 psi | Low tails, high cement = clay size |
| MED. | 50 - 100 psi | Med tails, med. cement = silt size |
| HIGH | > 100 psi | High tails, low cement = sand size |

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| | | | |
|---|---|---|---|
| FALCONBRIDGE MINE BACKFILL<br>CANMET SILL MAT PROJECT | DIP 0°<br>AZIMUTH 302.5° | LOGGED BY: A. BRADFIELD<br>SEPT. 23, 1986 | BOREHOLE<br>G4 |
| LOCKERBY MINE<br>2950 LEVEL | DRILLED BY CRAELIUS<br>DIAMEC 251/260 | | SHEET<br>1 OF 1 |

| SAMPLE NO. | FOOTAGE (FT.) | DESCRIPTION | REC. (%) | COMP. STR. (psi) | TAILS: CEMENT | MOI ( |
|---|---|---|---|---|---|---|
| | ROCK | | | | | |
| 249 | 25 | Drk, sand size, med. moist., very low str. & cem., all loose | | 25.1 | 17:1 | 20 |
| | | Sl. drk. than gr., moist., high str., | 95 | | | |
| | 30 | Sand to silt size, med. cem., no loose | | | | |
| 248 | | 6" loose section at 30' | 85 | 37.8 | 15:1 | 17 |
| | 35 | Drk, sand size, med. moist., very low str. & cem., all loose | 75 | | | |
| | 40 | | | | | |
| | | Grey, sand to silt site, low moisture & strength & cement, no layering | 30 | | | |
| | 45 | | | | | |
| | END | | | | | |
| | 50 | | | | | |
| | | OVERALL RECOVERY | 70 | | | |

REMARKS: PERSONAL CONVENTIONS USED

| | | | |
|---|---|---|---|
| AVG.RANGES FOR COMP.STR. | LOW 0 - 50 psi | SIZES: | Low tails, high cement = clay size |
| | MED. 50 - 100 psi | | Med tails, med. cement = silt size |
| | HIGH > 100 psi | | High tails, low cement = sand size |

| | | | |
|---|---|---|---|
| FALCONBRIDGE MINE BACKFILL<br>CANMET SILL MAT PROJECT | DIP 0°<br>AZIMUTH 325° | LOGGED BY: A. BRADFIELD<br>SEPT. 23, 1986 | BOREHOLE<br>G6 |
| LOCKERBY MINE<br>2950 LEVEL | DRILLED BY CRAELIUS<br>DIAMEC 251/260 | | SHEET<br>1 OF 1 |

| SAMPLE NO. | FOOTAGE (FT.) | DESCRIPTION | REC. (%) | COMP. STR. (psi) | TAILS: CEMENT | MOI (% |
|---|---|---|---|---|---|---|
| | | Gr.,silt to cl. size,high str.&cem.,low moist,no loose | | | | |
| | 5 | Drk. than gr, silt size, med. str & cem, low moist., all loose | 90 | | | |
| | | Gr, silt to cl size,high str&cem, low moist. all loose | 70 | | | |
| | | Drk than gr, silt to sand size,low str&moist,all loose | | | | |
| 252 | 10 | Lt gr, cl size, high str & cem, med. moist., no loose | | 202.7 | 6.8:1 | 15. |
| | | Gr, silt to sand size, low str & moist, all loose | 70 | | | |
| | 15 | Drk, silt to sand size, low str & moist&cem, all loose | | | | |
| | | Gr, silt to cl size, med str&moist,high cem, 30% loose | | | | |
| | | Drk, silt to sand size, high str cem, med moist | 70 | | | |
| | 20 | | | | | |
| | | | 50 | | | |
| 251 | | | | 180.9 | 9.8:1 | 13. |
| | 25 | Drk gr, sand size, low str & cem & moist, 50% loose | | | | |
| | | | 30 | | | |
| | 30 | Drk gr, sand size, low str & cem, med.moist. 70% loose | | | | |
| | | 1" bands of med. cem 30' to 31' | 40 | | | |
| | 35 | | | | | |
| | | Drk gr, sand, lo str & cem, med moist, 20% loose | 90 | | | |
| | 40 | Drk gr, sand size, low str&cem, med. moist, 20% loose. | | | | |
| | | 2 mm thick cl size brown coating on 50% of ore | 90 | | | |
| | END | | | | | |
| | 45 | End | | | | |
| | | OVERALL RECOVERY | 66 | | | |

REMARKS: PERSONAL CONVENTIONS USED

| AVG.RANGES FOR COMP.STR. | | SIZES: |
|---|---|---|
| LOW | 0 - 50 psi | Low tails, high cement = clay size |
| MED. | 50 - 100 psi | Med tails, med. cement = silt size |
| HIGH | > 100 psi | High tails, low cement = sand size |

| | | | |
|---|---|---|---|
| FALCONBRIDGE MINE BACKFILL<br>CANMET SILL MAT PROJECT | DIP -3°<br>AZIMUTH 325° | LOGGED BY: A. BRADFIELD<br>SEPT. 23, 1986 | BOREHOLE<br>P2 |
| LOCKERBY MINE<br>3125 LEVEL | DRILLED BY CRAELIUS<br>DIAMEC 251/260 | | SHEET<br>1 OF 1 |

| SAMPLE NO. | FOOTAGE (FT.) | DESCRIPTION | REC. (%) | COMP. STR. (psi) | TAILS: CEMENT | MOI (% |
|---|---|---|---|---|---|---|
| 254 | 5 | Grey, clay to silt size, med. strength & cement & moisture, 40% loose | 75 | 57.9 | 11:1 | 16. |
| 253 | 10 | Grey, clay to silt size, med. strength & moisture, higher than med. strength, 25% loose | 90 | 57.9 | 13:1 | 18. |
| | 15 | Grey, clay to silt size, med. strength & cement & moisture, 30% loose | 90 | | | |
| | 20 | Grey, clay to silt size, med. cement & strength, low moisture, 60% loose | 90 | | | |
| | 25 | Grey, clay to silt size, med. cement & strength & moisture, 60% loose | 30 | | | |
| | 30 | Grey, clay to silt size, med. cement & strength & moisture, 30% loose<br>End | 95 | | | |
| | | OVERALL RECOVERY | 78 | | | |

REMARKS: PERSONAL CONVENTIONS USED

AVG.RANGES FOR COMP.STR. LOW 0 - 50 psi SIZES: Low tails, high cement = clay size
MED. 50 - 100 psi Med tails, med. cement = silt size
HIGH > 100 psi High tails, low cement = sand size

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| | | | |
|---|---|---|---|
| FALCONBRIDGE MINE BACKFILL<br>CANMET SILL MAT PROJECT | DIP -3°<br>AZIMUTH 302.5° | LOGGED BY: A. BRADFIELD<br>SEPT. 23, 1986 | BOREHOLE<br>P5 |
| LOCKERBY MINE<br>2950 LEVEL | DRILLED BY CRAELIUS<br>DIAMEC 251/260 | | SHEET<br>1 OF 1 |

| SAMPLE NO. | FOOTAGE (FT.) | DESCRIPTION | REC. (%) | COMP. STR. (psi) | TAILS: CEMENT | MOIS (%) |
|---|---|---|---|---|---|---|
| | ROCK | | | | | |
| | | Gr., sand size, med. moist. & ceme.,high str. no loose | 20 | | | |
| 250 | 25 | | | 75.4 | 8.9:1 | 16. |
| | 30 | Lt. gr., silt size, dry, high cem., med.str. brittle, all loose | 25 | | | |
| | | | 0 | | | |
| | 35 | No recovery | | | | |
| | 40 | Drk black, sand size, very moist.,no str. & cem, all loose | 10 | | | |
| | | Gr.,sand size, med. moist., low cem. & str., all loose | 80 | | | |
| | 45 | Gr.,sand to silt size, med. moist. & cem., & strength, 50% loose | | | | |
| | END | | | | | |
| | 50 | | | | | |
| | | OVERALL RECOVERY | 32 | | | |

REMARKS: PERSONAL CONVENTIONS USED

AVG.RANGES FOR COMP.STR. LOW 0 - 50 psi SIZES: Low tails, high cement = clay size

MED. 50 - 100 psi Med tails, med. cement = silt size

HIGH > 100 psi High tails, low cement = sand size

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# APPENDIX F

# BLAST VIBRATION MONITORING RESULTS

## BLAST VIBRATION MONITORING RESULTS

### GENERAL MONITORING OBJECTIVES

1) To observe and evaluate the effectiveness of VCR mining; including attention to misfires, cross-initiation, sympathetic detonations and excessive vibration levels.

2) To provide additional insight into the effects of blasting in fill, including information on vibration levels, frequencies, durations, effects of cement content and vibration behaviour in fill compared to rock.

### INSTALLATION

A number of holes drilled through rock and fill (see Figures 23, 24) were used to locate geophones. The installation comprised:

1) In Fill - one triaxial velocity gauge array was grouted into a hole drilled in each of filled stopes alongside the pillar being mined at both the overcut and undercut horizons. Subsequent analysis of traces proved the installation and/or geophones in undercut station G1 was defective.

2) In Rock - a number of sites were used in access cross-cuts at the undercut elevation while the stope overcut was used at the upper elevation.

As the blasting horizon retreated upwards, monitoring shifted from the undercut to overcut horizons (generally).

### MONITORING: GENERAL RESULTS

From August 22 to October 10, 1986, the first ten blasts of 31-176 VCR stope were monitored using a Racal FM tape recorder and/or two Instantel DS-200 Collector Units. Some general remarks concerning the monitoring of each blast are now made.

Blast #1 - Simultaneous detonation of four decks of watergels in firing the dice produced relatively high vibration levels (over 76 cm/sec peak particle velocity (P.P.V.) in rock at 24 m). This exceeded the settings on the Racal FM tape recorder so that the maximum vibrations were not evaluated. The Racal also monitored a geophone location in the fill on the U/C location. These results indicated P.P.V.'s in the order of 71-76 cm/sec in fill while on the other side of the stope an Instantel recorder showed P.P.V.'s about 0.86 cm/sec. Subsequently, there existed some doubt about the reliability of the Instantels in observing "higher" vibration levels. Unfortunately, this was not substantiated until completion of the stope when a computer and hard copy facility (a graphics quality printer) became available.

Blast #2 - The Racal failed to record during the blast. The Instantels were located at the O/C in rock and the U/C in fill. They recorded a blast either 100 or 250 msec in duration whereas the design specified 1000 msec. As the entire blast was loaded with watergels, it appears likely that sympathetic detonation resulted in a very short and violent blast that exceeded the recorder's capabilities (specified as 25 cm/sec).

Blast #3 - Three recording units were used. Disparity was noted between Racal and Instantel observations from fill at the U/C horizon. Instantel results from the O/C in rock showed an unlikely maximum P.P.V. of 1.6 cm/sec. The blast again used watergel in all the holes so sympathetic detonations were noted in abundance.

Blast #4 - In an attempt to work within the capacity of the Instantel recorder, geophones were positioned in the U/C horizon at distances of 53 m from the blast. Unfortunately, increasing the distance necessitated positioning them so that a direct line of travel of blast wave from source to geophone was not possible.

The observed P.P.V.'s were minimal in magnitude (1-2 cm/sec) and indistinct.

Blast #5 - All three devices were used to record the blast from the U/C horizon. One Instantel was positioned 53 m away in an indirect line of wave travel. By this time, the blasting horizon had retreated to about 15 m above the U/C monitoring horizon and general travel distinctness was deteriorating for all units.

Blast #6 - The Racal was positioned at the overcut in rock and fill but due to the position relative to the blast horizon the shots were not all distinct. The Instantels were in the fill at the U/C horizon.

Blast #7 - Due to the run-down battery, the Racal did not record. One Instantel was placed at #1 Shaft due to some damage having been done there recently. The other was placed in the fill near the U/C and utilized the defective geophone array G1.

Blast #8 - The Racal was set up on the U/C horizon monitoring both fill and rock. An Instantel was connected to the geophones in fill at the O/C horizon. One unit produced a very poor trace and the other very low vibration levels (max. 3 cm/sec P.P.V.).

Blast #9 - The Racal was set up on the U/C horizon to monitor fill and rock. An Instantel monitoring in fill on the U/C produced a very poor trace while one on the O/C produced very low vibration levels (max. 4 cm/sec P.P.V.).

Blast #10- The Instantels monitored the fill in both U/C and O/C horizons.

## RESULTS AND OBSERVATIONS

The tangible results of attempting to provide information on vibrations in fill are summarized in the three accompanying graphs.

Figure 25 compares vibration levels between rock and fill based on equivalent scaled distances. In the first range, where rock P.P.V.'s were observed between 0-10 cm/sec, P.P.V.'s in the fill averaged 63% of the magnitudes of those in rock. Where greater magnitude vibrations in rock were recorded, this percentage lowered to 24%. In the range of the highest vibrations in rock (35-100 cm/sec P.P.V.), the vibrations in fill averaged only 10% of those in rock. On the basis of this data, the difference in vibrations between rock and fill is not a simple linear relationship, but depends on the magnitude and rate of release of blast energy and the monitoring distance from the blast. However, it is quite apparent that the difference in vibration levels between rock and fill is very significant.

Figure 26 describes the attenuations of blast vibrations in the two media. In this case, the rock consists of brecciated sulphides and granite gneiss whereas the fill is a generalization of 22:1 and 12:1 cemented hydraulic fill. The graph suggests that vibrations in rock tend to die out somewhat faster than in fill. The horizontal shift in the two regression lines is due to the bulk of the data for fill being observed from shorter distances to the blast. These data are all derived from Racal recordings.

Figure 27 indicates there may be differences in attenuation characteristics of fill depending on the cement content. The lack of data and the essentially lower scaled distances for the higher cement curve tend to decrease the reliability of the graph. All these data points are from Racal records.

Further observations of a more general nature are now given.

a) The dominant frequencies for fill were at least an order of magnitude less than in rock (30 Hz vs. 300 - 500 Hz).

b) Compression wave velocity is noticeably slower in fill than in rock (in the order of 20 msec later arrival time over 43 m distance in one instance).

c) Shot vibrations were of usual duration of 15-20 msec where no overlapping sympathetic detonations occurred.

## FINAL COMMENTS

A Vertical Crater Retreat blast is difficult to monitor, especially when the stope is great in height (58 m in this instance). A more ideal situation to compare vibrations in rock vs. fill would take place in bench-blasting a stope with fill on one side and rock on the other. The nature of a VCR stope entails most of the blasting being done on a horizon not close to either overcut or undercut where monitoring is easiest.

Monitoring a new mining method, especially one which is developing (and changing with every blast) does not provide ideal situations for producing consistent data. When sympathetic detonations occur, the exact identity (and location) of the shots firing is uncertain, producing dubious scaled distances.

Well tried and tested monitoring equipment is mandatory for producing accurate data. In this instance, the Instantel recorder had been tested but only at much lower vibration levels.

# APPENDIX G

## STATE-OF-THE-ART REVIEW ON BACKFILLING TECHNOLOGY

# STATE-OF-THE-ART REVIEW ON
# BACKFILLING TECHNOLOGY

## 1.0 INTRODUCTION

An in-depth state-of-the-art review on backfill technology in a number of mines worldwide, was carried out prior to commencing the project. The main objectives of this review were to:

a) Assess suitable drilling techniques that may be applicable for recovery of core in cemented and uncemented hydraulic and rock backfills.

b) Evaluate the general backfilling practices in relation to mining methods, orebody geometry, and general working environment.

c) Identify areas for potential future research on behalf of CANMET and the Ontario Mining Industry.

The state-of-the-art review, included a literature search and visits to selected mines in Canada and abroad. Specifically, the following was carried out:

1. A review of existing mining practices in Canada and worldwide through a literature survey.
2. Visits to selected mines in Ontario notably, Timmins and Hemlo Lake areas.
3. Visits to Australia and Europe to examine backfilling practices first hand.

The above state-of-the-art review was carried out jointly with another CANMET project being completed under sub-contract to Dome Mines Limited (Dewatering Schemes for Mine Backfill - DSS Contract #OSQ 85-00298). Part of the review work has, therefore, been summarized in the report on the sub-contract project to Dome. The highlights of the review are presented below. In this section, emphasis has been placed on drilling techniques in backfill.

## 2.0 LITERATURE REVIEW

A number of published papers were reviewed on the mines in Canada and abroad. Emphasis was placed on operations in Australia and Europe because of the extensive research effort that has been carried out in these countries with regards to backfill. The main conclusions from this review are as follows:

1. Most of the research work on backfill has been carried out in the laboratory. Only a few in situ studies on backfill behaviour have been conducted. This is partly because of the difficulty associated with the accessibility and drilling in fill, and also partly because of the expenses involved in conducting such work.

2. There is a wide division between laboratory studies and available field results. In most instances field data indicate problems with segregation and quality control. In addition, much of the information seems to indicate that, in-situ fill behaves much better than predicted by laboratory data. It appears that, horizontal and vertical compression and confinement of the fill, enhances its properties by improving overall densities and reducing the void ratios.

3. In the case of mines that rely on backfill for regional and local support, there seems to be a distinct preference for cemented waste rock and high density tailings fill over other types of fill. The use of hydraulic fill is generally limited to either cut and fill mining or to operations where the primary role of the fill is to backfill large voids for eventual pillar recovery.

4. Overall experience in Australia, Europe and Canada with drilling and sampling of backfill is extremely limited and has been very poor. Most companies report recoveries of less than 20-25% at a generally prohibitively expensive cost for drilling. Many of the mines seem to have abandoned any hopes of being able to reliably core backfill in situ with current available technology.

The overall impression from this literature review was that, CANMET's projects on the design of sillmats and in situ backfill studies were timely. It would appear that this project will have implications not just for the Ontario Mining Industry but for underground hard rock mines worldwide.

## 3.0 CANADIAN OPERATIONS

Backfill design in Ontario and Canadian mines appears to be based on experience. Over the last few years, there has been a definite trend towards cemented waste rock fills (as opposed to cemented hydraulic fill). This reflects a need to obtain a much stiffer backfill at a lower cement cost, the desire to effectively dispose of waste, and the requirement to provide regional support underground at optimum cost.

Falconbridge's Kid Creek Operations have conducted much of the work in this area and are regarded as Canadian pioneers in the use of cemented waste rock. Although many of the mines have simplified the backfill procedures from those used at Kidd Creek and therefore made the system less expensive and simpler from an operational point of view, the basic concepts have remained unchanged. Waste rock is mixed with a cement slurry and dumped into the stope. The waste rock is often crushed to control its material size distribution to ensure percolation of the cemented slurry into most of the voids. In some cases, as in Geco Mines in Manitowadge, Ontario, cemented tailings slurry is used after the stopes have been filled with waste rock. The cement tailings slurry in a 1:30 ratio, appears to percolate through the waste rock over vertical heights in excess of 150 m in some places.

Most of the recent mines brought into production in the Hemlo area have opted for cemented waste rock. Both Noranda and Lac Minerals have selected this sytem. The waste is quarried on surface specifically for fill. The design criteria is largely based on laboratory results to meet certain strength characteristics and the experience gained at Kidd Creek and Geco Mines.

Efforts to recover core from backfill have been limited. Most of the work has previously been done at Falconbridge's Sudbury Operations and at Inco's Operations in Sudbury. The main objectives have been: recovering in-situ core, to provide an accurate hole for instruments (such as extensometers and pressure meters) and to conduct quality and cement content tests on the core. A major project was carried out by Trow Limited under contract to Falconbridge between 1978 to 1980. The diamond drilling was conducted at Strathcona Mine and recoveries were lower than 25%. The main conclusions of the study were that present technology does not permit good recovery of core from backfill operations. It was also pointed out that the fill samples showed a wide variance in cement content. On this basis, more research into better techniques for drilling into backfill was recommended.

## 4.0 AUSTRALIAN MINES

Visits were made to selected Australian mines as follows:

1) Mount Isa
2) Broken Hill
3) Olympic Dam
4) Que River and Mount Lyell in Tasmania

Mount Isa was selected because it has perhaps the most comprehensive and extensive backfill program in the world. The mine uses six different types of backfill, and over the years has carried out extensive research to optimise cement consumption and overall backfill technology. Broken Hill and mines in Tasmania have been using conventional hydraulic fill for a number of years. Because of their experience with pillar recovery operations, a visit was considered essential. Finally, Olympic Dam was selected because it is the newest, large base metals operation in development in Australia. Falconbridge felt it would be important to identify what new technologies in backfill are being incorporated into an operation such as Olympic Dam.

Two common elements in these mines were evident:

1) Dilution from backfill typically ranged from 10-20%. These figures are common in Canadian mines.

2) Very little in situ drilling for core recovery had been carried out. Most of the basis for design was from laboratory test data and field experience.

Most of the Australian experience appears to be at Mount Isa and this has set the trend for other mines. The extraction of lead and copper ores at Isa Mine makes large stope voids underground. To maintain stability of the underground workings, each year substantial amounts of material must be placed to fill these voids. The materials used are:

Dry Fill

Graded rock from a pit is used in the copper orebody. It is choke fed from the surface through fill passes and distributed by conveyors. The majority of the 3 million tonnes of graded rock placed in any one year is mixed with cemented hydraulic fill (CHF) in a 2:1 rock:CHF ratio.

Crushed waste and pit fines are used to fill stopes in the upper part of the mine. Material is fed through a series of fill passes. After dry filling, the stopes are tight-filled with wet fill.

Hydraulic Fill

About 57 weight percent of the tailings is recovered as fill, the rest is pumped to the tailings dams. The three combinations of fill used are:

- Hydraulic fill (HF) - used primarily in cut-and-fill stopes.
- Cemented hydraulic fill (CHF) - consists of a mixture of HF with cement and finely ground copper reverberatory furnace slag (CRFS). A typical mix containing three weight percent cement and six weight percent CRFS is equivalent in strength to a six weight percent cement mix, and is cheaper to produce. CHF on its own is used mainly in the lead open stopes.
- Cemented rock fill (CRF) - is a mixture of CHF and graded pit rock (siltstone). This material is used in the copper orebody in stopes alongside which pillars are to be recovered.

Hydraulic fill is fed through six fill holes and is delivered underground at an average rate of 215 tph. CHF is placed at an average rate of 230 tph.

Most of the research into backfilling has focused on the following areas:

1) Development of numerical modelling methods to more reliably simulate in-situ fill behaviour. This work is similar to that being carried out by CANMET and a number of mines in Canada.

2) Observation of placement conditions and actual performance based on photography and in situ monitoring.

3) Laboratory testing of cemented fill samples made from different combinations and under varying conditions to assess mechanical properties and expected performance underground.

4) Fill quality monitoring from the fill plant down to actual delivery in the stope.

Most of the research at Mount Isa has been towards reducing cement content and optimising placement methods. Very little work has gone into examining the regional stability benefits of backfill and design of sill mats.

## 5.0 EUROPE

The main focus of backfill technology of interest to this specific project in Europe was identified to be with Outokumpu Oy in Finland. Outokumpu's Pyhasalmi Mine carried out a major project on freezing uncemented hydraulic and waste backfill underground and recovering a pillar adjacent to the frozen backfill. The value of the adjacent pillar justified the freezing operation and its subsequent recovery. The following are the highlights:

A series of holes, approximately 1.5 m apart, were drilled into the backfill and a mobile refrigeration plant used. By circulating brine in the holes, the unconsolidated waste in tailings fill was successfully frozen while the pillar in the centre was recovered. The freezing system used is an indirect one. The liquid in the compressor is Freon 22 and a solution of calcium chloride circulates in the cooling tubes. The equipment was mounted underground near the stopes to be frozen. Temperature sensors were mounted to ensure that the fill temperature was below freezing.

The total project costs ran as follows:
Development work 14%,
Drilling and installing of the cooling tubes 45%.
Mounting the equipment 23%,
Maintenance of equipment 8%,
Energy 8%,
Control of the equipment 2%.

The most difficult and expensive part of the work was drilling and installing of the tubes, mostly because of the non homogeneous material. The material varied from fine grade sand to 2 $m^3$ waste rock blocks. The extra cost of ore recovered using freezing is estimated at $10 to $20 per tonne. Drilling was carried out using a double tube diamond drill core barrel. The holes had to be cased throughout, resulting in a per m cost in excess of $150. This is well in excess of typical costs of $50 to $6/m in Canadian Mines.

## 6.0 CONCLUSIONS AND FOLLOW-UP

The main conclusions from the state-of-the-art review are as follows:

1. The extent of drilling in backfill for core recovery is extremely limited. Most Canadian companies are using state-of-the-art technology and, in some instances, are well ahead of other mines in different parts of the world.

2. The leader in backfill and technology is Mount Isa Mines in Australia. In spite of the fact that the extent of drilling for core recovery is limited, the research at this mine has been extensive. Although much of the work has been done with government funds, especially with the CSIRO geomechanics division in Australia, the focus of the work has been Mount Isa's operations.

3. The unique method for freezing backfill used at Pyhasalmi Mine in Finland is interesting and may well prove cost effective in certain areas.

NOTE: Details of the mines and backfilling techniques mentioned are available upon contact with Falconbridge Limited's Sudbury, Ontario Operations.